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THE JOURNAL

OF THE

CHEMICAL, METALLURGICAL & MINING SOCIETY

OF SOUTH AFRICA.

VOLUME VIII.

JULY, 1907 — JUNE, 1908.

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JULY, 1907—JUNE, 1908.

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THE JOURNAL
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Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

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Vol. VIII., JULY, 1907. No. 1.

Proceedings
AT
Ordinary General Meeting, July 20, 1907.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, July 20th, Prof. J. Yates (President), in the chair. There were also present :—

13 Members: Dr. J. Moir, Messrs. T. L. Carter, J. Littlejohn, R. G. Bevington, W. R. Dowling, G. O. Smart, A. Whitby, H A. White, Prof. J. A. Wilkinson, W. Beaver, H. D. Bell, B. V. Blundun, A. A. Coaton, M. H. Coombe, G. A. Darling, N. M. Galbraith, F. N. Hambly, W. H. Jollyman, J. A. Jones, J. Kennedy, G. A. Lawson, H. Lea, C. W. Lee, C. D. Leslie, W. P. O. Macqueen, G. Melvill, H. H. Morrell, P. T. Morrisby, W. D. Morton, T. T. Nichol, F. B. Ogle, C. F. Parry, J. F. Pyles, O. D. Ross, C. B. Saner, S. Shlom, W. H. Stout, H. Taylor, J. A. Taylor, H. Warren, John Watson and J. O. Welch.

12 Associates and Students: Messrs. S. J. Cameron, J. Chilton, C. L. Dewar, W. J. N. Dunnachie, J. H. Harris, C. B. Hilliard, E. Homersham, R. Lindsay, C. Toombs, A. Waugh, H. Weldon and L. J. Wilmoth.

12 Visitors and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as published in the *Journal*, were confirmed.

New Members.

Messrs. Melvill and Saner were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been duly elected, as follows :—

Evans, Thomas, Knights Deep, Ltd., P. O. Box 143, Germiston. Assayer. (*Transfer from Associate Roll.*)

Greer, Jr., James Courtney, Knights Deep, Ltd., P. O. Box 143, Germiston. Chemist.

Jackson, Richard, Treasury G. M., Ltd., Cleveland. Shift Boss.

Jolly, Harold R., Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Chief Assayer. (*Transfer from Associate Roll.*)

Scarf, Henry A., Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanide Foreman.

The Secretary announced that the following gentlemen had been admitted Associates by the Council since the last general meeting.

Angus, William Hay, Knights Deep, Ltd., P. O Box 143. Germiston. Foreman Amalgamator.

Bayley, Charlton F., Knights Deep, Ltd., P. O. Box 143, Germiston. Mill Foreman.

Jones, G. J., Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Technical Chemist.

Powter, Harold Bowen, Knights Deep, Ltd., P. O. Box 143, Germiston. Engineer.

Wilson, Peter, Knights Deep, Ltd., P. O. Box 143, Germiston. Mill Foreman.

INAUGURAL ADDRESS.

By Professor John Yates (President).

I have to thank you for the honour you have conferred on me, an honour which I fully appreciate. The leading of this, one of the premier societies of South Africa, is no light responsibility, and probably never was the responsibility greater than at the present juncture when both the gold industry and the State are in such an unsettled condition consequent on the repatriation of the Chinese, the strike of our miners, the coming into power of the new Government with its many possibilities, and, above all, the rude shock which has been experienced by all those who, having the permanent welfare of the country at heart, realise for the first time that the average grade of the Rand is merely 30s. per ton milled ; I question whether even now the full significance of this last point and its vital incidence on the whole economic framework of the country has yet been fully grasped. Never before have our economic conditions been so disturbed, depression been so acute, or ruin hovered over so many, and it but increases the bitterness of the position when we find that South Africa is alone in the pit of depression and that the world generally is revelling in a surfeit of abundance. But our industrial depression should merely act as an incentive to all in South Africa to put their shoulder to the wheel and make a determined effort to help the country along, and I feel sure that what we, as a Society, can do in this direction we will most willingly perform.

Dr. Moir has drawn your attention to the fact that I am the first mining president the Society has had, and it is somewhat of a coincidence that I was largely instrumental in adding mining to the Society's title and work. This broadening of our scope was, as you will doubtless remember, not effected without opposition from a section of our members, but even they must now be convinced of the wisdom of the change—a mining society was much needed on these fields, and I submit that, judged by their contributions, our mining members have done right well, and that our reputation as the Mining Society of South Africa is now firmly established.

Mr. Johnson has kindly left me to review the past year's work, and as an old member—Mr. Rowland tells me I joined in 1896, when the membership was only 91—I have a duty, a pleasant duty, to perform, and that is to take advantage of my long acquaintance with the members and on your behalf, give thanks to just a few of those to whom thanks are due. Since the war you have had as your presidents, Williams, Pearce, Caldecott and Cullen, and this "Old Guard," ever watchful of your interests, Mr. Johnson has just joined, a worthy associate for our other veterans. Of your indebtedness and the indebtedness of the mining and metallurgical world to these men I do not purpose speaking except to say just a few words on Mr. Johnson's labours during the past year. He is, I think, even an older member of the Society than myself; he had a long apprenticeship and did much solid work before taking the chair, the onerous duties of his appointment in no way interfered with his devotion to the interests of the Society, and for this and his unfailing courtesy he has well earned our thanks. Of our Mr. Rowland so many nice things have been said from time to time that I will content myself with remarking that he deserves them. Next we come to a veritable corner stone of the Society, our esteemed Hon. Treasurer, Mr. Littlejohn, who after many years exposure to the stress of our financial matters presents to them as smiling and unruffled a front as ever. Of Dr. Moir and Mr. Bradford, who were associated with me as Vice-Presidents, it is a pleasure to place on record the good work they have done; Mr. Bradford especially has, as we all know, had a year of strenuous labour, working, as he well knows how to work, in the interests of the industry and the country. In concluding this very brief review, I think I may well say of the *personnel* of the present council that its constitution is a matter for congratulation both for the Society and myself.

I trust that you will continue to strive to lengthen our membership roll, and I wish to extend a cordial invitation to all of you to come forward with papers and take part in our discussions and thus benefit both yourselves and your co-workers. As a Society we are noted for our discussions: their breeziness has contributed in no small measure to the making of the Society and many of its members, yet in this connection I need hardly remind you that authors, even when we may differ from their views, are entitled to a full measure of courtesy at our hands in return for their expenditure of time and thought on the Society's behalf.

I believe I am expected to-night to review the past, comment on the present, and forecast the future.

The Past Year.

The past year will be recorded in South African history as one of acute depression, a year of losses for financiers, shareholders, and tradesmen alike, of empty houses for landlords and of relief works for the needy. So far as our Society is concerned it is not necessary that I should reiterate what you have already perused in our Annual Report.

The papers contributed were of a very varied and instructive nature, and the authors are to be congratulated on the standard they attained, the mining papers being especially deserving of commendation. There is, I am pleased to note, a tendency among our members to deal with details of plant and practice; I hope that this tendency to bring forward little departures from ordinary routine will grow and that members will be good enough to convey to their fellow workers, through our transactions, ideas which may prove of service directly or indirectly.

The year has in many ways been a remarkable one for mining and metallurgy, and on the Rand many developments worthy of note have occurred. Thus, a 300 stamp mill with steel battery framing has been ordered for the Simmer Deep, Ltd., the stamps will be 1,750 lb., probably the heaviest gravity stamps in the world. The crusher station of this installation will also be steel framed. The Gold Prince Mill, Colorado, is an example of a large mill (100 stamps) built almost entirely of steel in the manner proposed for the Simmer Deep, Ltd., and having steel battery posts.

Several small improvements have been effected in our tube-mill practice and very little has been heard during the year of the Wheeler pan as a supplementary crusher on these fields. Steel framing for shafts in being introduced in the Kleinfontein group and our latest seven-compartment shafts are being equipped with a permanent repair cage in the pump and ladder way. The mono-rail is making headway in connection with our underground transport and is about to be installed on the Rand mines subsidiaries. The unsettled state of the unskilled labour question gave rise to a boomlet in connection with small machine drills, but once again expectations have not been realised and we still await a really simple and efficient mechanical substitute for hand drilling in small stopes.

Good work has been done by the Safety Catch Commission, and from the multitude of designs submitted to them three have been proved to possess outstanding merit, viz., the Garvin, the Schweder, and the Undeutsch ; but whilst the merit of these designs compared with the old standards is obvious, it is noteworthy that none of the three are free from that greatest of all objections to such safety appliances, namely, the liability to come into play when not wanted, but, at any rate, it is satisfactory to note that the damage likely to arise from such uncalled for action is greatly reduced in these new types.

The year has advanced our knowledge of explosives considerably and we have come to realise that nitro-glycerine compounds have little respect for formulæ, that perfect detonation is the exception and not the rule, and that our blasting operations are almost invariably attended by the evolution of those objectionable gases, NO and CO, and, this being the case, our miners would do well to assume that these gases are present after every blast. The subject of the ventilation of our mines has been very much in evidence during the year, and though Dr. Moir's paper anent this raised a storm of protest the interesting controversy has done a certain amount of good inasmuch that it has attracted attention to the absence of ventilation in certain working places underground, a state of affairs which, however, also occurs on other mining fields. To this extent, therefore, the discussion was advantageous, but I am afraid that otherwise the paper and discussion have unfortunately given rise to much unjust prejudice against the quality of the ventilation on our mines generally.

Our working horizon is steadily descending, our deepest mine, the Simmer West, Ltd., being about 4,500 ft. deep, but we are, in this matter, behind the copper mines of the Lake Superior district which are at present sending ore to surface from over 5,000 ft. vertical and apparently see no technical difficulty to delving to 10,000 ft. vertical.

Turning now for a moment to other mining countries among noteworthy events of the year is the starting up of a slime plant on the Homestake Mine to treat ¾ dwt. slime, a feature of the plant being a filter press with an automatic sluicing device by means of which the press can be quickly emptied of its contents without being opened. The crude and laborious method of discharging filter presses hitherto obtaining has been one of their greatest drawbacks, and it is to be hoped that the Homestake Co. has satisfactorily overcome this disadvantage. Considerable ingenuity is at present being concentrated on the treatment of slime by other methods than decantation, and among the ideas being tested on other mining fields are the Moore filter basket and the Ridgway machine. The latter consists of a number of horizontal circular plates revolving around a central vertical tubular column to which they have a pipe connection ; each plate has an under filtering surface and dips through a portion of its revolution into the slimes pulp ; a cake is formed by the application of a vacuum and the plate continuing its revolution next finds itself in a water bath where it is washed and the cake subsequently dislodged ; this is, I say, ingenious, but on the Rand the decantation process leaves little to be desired.

Magnetic separation is making great strides in Sweden where iron ores well suited for this treatment abound, and according to Prof. Peterssen, the Professor of Mining at Stockholm, there are now 21 plants in operation and concentrates containing 63 to 68 per cent. iron can without difficulty be produced from a crude ore with from 25 to 30 per cent. of iron if crushed fine enough, the loss of iron in the tailings being comparatively small.

Flotation processes have been largely tested during the year and they promise to be of considerable assistance in the treatment of certain ores of base metals.

Cornwall, that birthplace of British mining, which has so long been content to live in the past is at last waking up and is throwing out its historic buddles and frames to make way for the Wiltley and such like approved modern appliances. The magnetic separator is also doing good work on the Cornish tin ores separating the tungsten and materially augmenting profits, the production of tungsten ore from Clitter's United and East Pool mines in 1907 being valued at £8,174.

A matter of interest to the mining world has been the development of the cobalt deposits of Ontario, Canada, these ores carrying silver, nickel, and arsenic, in addition to cobalt, and being richer than the New Caledonian deposits. The year was a boom year for mining and it witnessed remarkable increases in mineral production and consumption. The Transvaal, Australia and United States of America together produced over 70 per cent. of the world's output of gold which is now about £80,000,000 per annum. The world's silver production exhibited little change, but over 700,000 tons of copper, 900,000,000 tons of coal, 100,000,000 tons of iron ore, 13,000,000 tons of salt, 1,000,000 tons of lead, etc., were mined, these figures giving some idea of the magnitude of mining operations. The last return of the world's miners shows that there are 3,644,416 distributed in the different countries, as follows :—

Great Britain and Ireland ...	974,634
Germany	814,352
United States of America ...	607,069
Russia	344,255
France	322,536
Austro-Hungary	225,371
Japan	163,530
Other Countries	192,669

Legislation has been proposed in the United States for the prevention of fraud in connection with the promotion of mining enterprises, the idea being to make misrepresentation a felony and fixing heavy penalties. But the public is itself largely to blame for the often unscrupulous methods of company promoters : so long as people, in their mad rush to make money, blindly accept the reports of men who are not qualified to report or speak on mining matters how can you expect otherwise than that a certain class of people will make a business of trading on this weakness : ways of evading the penalties of the law can generally be discovered, and any attempt at saving the public from itself in this matter by legislation looks well nigh hopeless in view of the fact that some people do not think the possession of a report is a necessary preliminary to investment in a property. It is of little use attempting to reason with many of the investors in mining stock, they may be sane men in their own business but the glamour of a possible bonanza mine is too much for their mental equilibrium.

As in 1905 the Transvaal last year was easily the world's greatest gold producer, the output being approximately £24,000,000 sterling : the United States and Australia were respectively second and third on the list, and it now seems improbable that we will be deposed from our premier position for many years to come. Our gold industry is magnificent in its proportions : that it should at the present time be almost the only support of the country is the Transvaal's misfortune, and because of this dearth of other revenue producing industries it is gratifying to hear that it is proposed to manufacture cyanide locally : our mines now spend over £300,000 per year on this commodity.

THE PRACTICE OF TO-DAY AND THE POSSIBILITIES OF TO-MORROW.

To speak of the past is easy but attempts to outline the future are often seen in the after light of developments to be merely caricatures. But so far as metallurgy is concerned I think I may safely say that its development has in most cases exceeded expectations, that what was in past years hoped of it has been very often surpassed by actual accomplishments, and it may be that this pleasing trait will be maintained in the future. I have often thought that we would do well to thoroughly reconsider our impressions and practices every few years. Our practice at any time is based upon the particular conditions existent at that period, and as the conditions change so should our methods if we are to reap the full benefit of economic and other developments. Metallurgy especially has been advancing so rapidly of recent years and working conditions in the Transvaal exhibit such a betterment compared with pre-war days that we cannot be too watchful of events or too critical in our inspection of our own work ; propositions which would have been absurd in the old days may now, owing to the changes which have occurred, be feasible and payable schemes.

I shall not be at all surprised to see a few years hence, economic conditions permitting, 800 stamps or more under one roof on these fields. Such a mill need not necessarily be attached to one mine : for instance, three large adjoining deep level properties, each of 400 stamp capacity, and each independent so far as shareholders interests are concerned, might advisedly have a joint 1,200 stamp mill with suitable workshops, power station and staff; this is what the unrivalled expanse of our gold beds calls for, and this operating on a large scale, a scale commensurate with that of the deposit, is what will have to be done if we are to work these reefs of ours to the depths they should be worked. We want larger properties, great joint mills and a cheapening of our surface installations and our working costs. Turning now to detail, the use of the rotary sand distributor at work on the Langlaagte Deep, Ltd., is being attended with an improvement in extraction and we may expect to see it thoroughly tested. It is improbable, I think, that any further cyanide plants of the superposed type will be erected : the plants of the future are more likely to have only a few catching tanks but a liberal number of treatment tanks, all more or less at one level, transference of the sand being effected by conveyor belts. When considering mechanical appliances, like the Blaisdell excavator, we have to bear in mind the cheapness, compared with most other mining fields, of our unskilled labour, and though the machine mentioned works well, I doubt that it can be installed in every case with advantage. Our tube mills are doing good work and have proved a friend to the shareholders by increasing our profits. Our stamps are, as I have already mentioned, taking on more weight year after year and I am inclined to predict that we will see 1 ton gravitation stamps and even larger. Compared with the giant steam stamps of the Lake Superior Copper mines with their capacity of 300 to 500 tons per stamp per 24 hours our stamps are but toys, and whilst I fully appreciate the difference in the quality of the product in the two cases I cannot help asking our metallurgists and engineers whether, under the conditions at present obtaining on the Rand, steam stamps are not at least worthy of consideration; I merely put this as a question you will observe. True progress is often hampered by conservatism on the one hand and ill advised attempts to advance on the other: it is for us to try and avoid both of these and I think it unfortunate that the continuous slime process has proved unsatisfactory on the New Goch because the failure may deter our engineers and financiers from venturing on any new lines unless absolute safety is assured, and this is often very difficult to guarantee in advance. The idea of all-sliming is in abeyance for the present, but with cheaper power and supplies we may expect a recrudescence of arguments in its favour: the new Usher-Adair process which has, I understand, passed trial runs satisfactorily will, it is to be hoped, cut down the present capital charge for slime tanks somewhat.

Concentration by Frue vanners on the Rand is about to become a thing of the past, the Langlaagte Estate having decided to discard them and fall into line with our general practice. The Wilfley table is still at work on the East Rand Proprietary mines and it remains to be seen whether its undoubted superiority to the vanner will enable it to hold its own. The treatment of residue dumps *in situ* has not been sufficiently successful to induce more than a very few mines to try it.

Relative to our mill construction the next few years will probably see timber battery posts superseded by steel, as the increasing weight of our stamps is likely to turn the scale against fear of trouble arising from crystallisation of the metal.

Concerning the advance of the metallurgy of metals other than gold I would call your attention to the recent presidential address of Prof. Gowland before the Institute of Mining and Metallurgy, London, wherein he deals with the present and future of the metallurgy of copper, and also to a recent paper on technical education by Prof. R. H. Richards, read at Halifax before the Mining Society of Nova Scotia, in which, in the course of showing how closely bound together are technical education and industrial advancement he spoke as follows:—"At about this time (1870) the American scientific schools began turning out their graduates and the Americans who had obtained similar training abroad returned. These men brought in a new leaven, and the resulting increase in the work of a furnace is represented by the following:—

In the year 1870 the charcoal furnaces then in use had a capacity of 10 tons each per 24 hours, a height of 39 ft., and a width of hearth of 2 ft. 6 in.

In 1904 the capacity of these furnaces had been increased to 160 tons, their height raised to 70 ft., and the width of hearth increased to 8 ft. 8 in.

In 1870 the anthracite iron furnaces in use had a capacity of 25 tons and a height of 40 ft., which, in 1904, had been increased to 465 tons per day and 100 ft. in height.

Iron furnaces burning coke had a capacity of 207 tons and were 75 ft. high as late as the year 1897, but in 1904 the latest furnaces had an output of 600 tons per day and stood 100 ft. high.

The same wonderful progress is true of the Bessemer process. When it was first introduced into the United States about 1868, two converters could work five charges of 5 tons each, 25 tons in all, each day. Now, through the improvements brought in by the introduction of scientific methods, two converters can make 900 to 1,000 tons of steel a day.

The story of copper furnaces has been the same. The reverberatory furnace of 1878 had a hearth of 9 ft. x 15 ft. and worked 12 tons in 24 hours. The Argo, Colorado, reverberatory now has a hearth 19 ft. x 112 ft. and works 350 tons.

The lead furnaces of 1870 were 24 in. sq. in section, and smelted 12 tons in 24 hours. Now the cross sections are 48 in. x 164 in., and smelt 160 tons a day.

Such extraordinary development as this came, not without resistance from conservatism, as the direct effect of the application of science and the scientifically trained mind to the subject in question."

Coming back to the Rand and its mining practice, steel collar sets for our shafts are likely to become popular because of their great permanency compared with timber and the innovation at Kleinfontein in the way of steel sets throughout will be watched with interest. Our shafts are increasing in sectional area as we go south to the deeper zones and the irresistible demand for less capital outlay and for maximum profits will, of course, result in their number, relative to a unit area, being reduced to a minimum. The deeper we go the more costly will our development become because, among other things, of the increasing heat and the longer time occupied in getting to and from ends, etc., and for these and other reasons I foresee the laying out of our deeper levels with longer backs in conjunction with mechanical transport of the ore in the stopes. On hand versus machine stoping our experience on these fields has demonstrated, in my opinion, that we should use hand drilling in stopes under 5 ft. wide, and $2\frac{3}{4}$ in. and larger drills in stopes over this size, and my own experience with "baby" drills was such as quite convinced me personally that there was little scope for them on the Rand.

On this field we have been more than once accused of indifference to labour saving devices, but, as a matter of fact, there is little in the shape of labour savers that we have not tried, some have successfully stood the test of practical application and have been adopted, others have failed ignominiously when put to the trial and have therefore been cast aside, and I believe that out of deference to public opinion, and with a whole-hearted desire to economise labour to the full, our engineers and managers are responsible for our having labour saving devices at work at the present day which it would benefit us to relegate to the scrap heap. And after all, as has already been pointed out, these mechanical contrivances touch our labour army very slightly, and we must remember that both white and coloured are affected and that labour savers do not necessarily mean a higher ratio of white to coloured labour, nor yet increased profits.

And now I come to another subject which has rightly assumed great prominence of late, and which I introduce to you without apology, for it is a matter in which this Society and the kindred Societies of Johannesburg should take a deep interest :—I refer to the subject of education. This Society has, in its time, done much educational work ; twelve years ago it was educating our cyanide and mill men, later on it took upon itself the education of our mining men. Its duties as an educational factor have largely passed to the Transvaal University College, with which I have the honour of being connected, but even now the Society is still an educational power in its own sphere of work, and it can, I think, be looked upon as a finishing school where all who care to may come and hear discussed the problems of our great industry. The Society is rightly noted for its instructive discussions, and I trust that our workers will come forward in the future, as in the past, and give of their experience and their knowledge for the good of the community.

Germany, the United States of America, and the United Kingdom—the leading commercial countries of the world—have long realised how vital to a country's prosperity is sound and thorough technical education, and money is being lavishly spent by them in this connection ; experience has taught them that it is an investment which will bring an ample return, and that a parsimonious policy in connection with a nation's educational establishments will surely and quickly result in a fall in that country's status. The Royal School of Mines, London, appreciating the necessity of teaching their men the practice, as well as the theory of their profession, has decided to spend a large sum of money on an ore dressing and metallurgical laboratory which is to be equipped on the most advanced lines, and a scheme has been evolved by which a number of their students are sent either to Australia, or these fields for practical work. Speaking as a man who has had the good fortune to pass through well nigh every phase of my profession, from the lower rungs to the top of the ladder, I cannot emphasise too much the importance of the rising generation being taught the practical application of theory and principles before they are turned on to the world to do battle for themselves, and more especially does this hold good in the case of the mining profession the circumstances surrounding which are undoubtedly exceptional. So far as the mining course of the Transvaal University College is concerned, there is, in some respects, little to complain of, in fact, we have, in certain directions, advantages which are unrivalled, but we have as yet no provision for the practical teaching of ore dressing, nor yet of the metallurgy of the base metals, but as equipment for this work is an essential feature of every mining school, and as nature has not been niggardly to South Africa in the way of base metals, it is to be hoped that this important mining country will see the wisdom of providing such equipment as will enable South Africa's metalliferous wealth to be worked by the men of South Africa. It is a sign of the times that there are two schemes afoot on the Rand for the practical training of miners, and these schemes are in no small measure due to a member of this Society, Mr. Bradford. By the one scheme adults are taken and taught machine drilling only. The other scheme is designed to give the young men of the country a thorough insight to underground practice with a view to making them expert breakers of ground. It is proposed that both classes shall be encouraged to attend the lectures at the Transvaal University College, which I think you will agree with me is wise, for, as you know, the close proximity of a scorching hot furnace is not the best spot to explain the reactions going forward therein, nor are our underground workings always the most suitable place wherein to expound our mining practice. The idea of devoting a mine exclusively to the training of white miners is naturally of special interest to me, but so many considerations are involved that I would rather be excused discussing it this evening. That we are badly in need of trained workers no one denies, that our miners as a body are inefficient is now generally admitted, but let us above all things be careful not to do injustice to the many first-class men among them who are fit to work alongside the best in their profession. The inefficiency is very largely with those who have more or less drifted into mining, men, strangers to the work, but welcomed by our mine foremen in the absence of trained men. Unfortunately, these men have in many cases been satisfied to learn the bare routine work and draw the very satisfactory pay attached thereto. In other cases the absence of facilities for study has undoubtedly contributed to the low standard of efficiency, but the evening lectures now open to all are doing away with this disability, and the fact of such numbers of all grades already availing themselves of the opportunities afforded is decidedly encouraging, and testifies that we have many ambitious men in the ranks who are determined to know their profession thoroughly and rise in it. In connection with this subject of education, a great and far-seeing business man has bequeathed the sum of £200,000 for the founding of a University in the Transvaal,—I refer to the late Mr. Feit.

So much for the education of the white man, now for the industrial training of the native. I am a great admirer of the native races of South Africa, and they are undoubtedly one of the country's greatest possessions; the Kafir is not good at thinking, but properly handled he is obedient and willing to a degree, and I do not consider that we have so far availed ourselves of anything like his full capabilities. With a little selection and training, I am, for instance, convinced that our Kafirs would be capable of doing everything in connexion with machine drilling except selecting the benches, locating and pointing the holes, and charging and firing, and some of the experienced boys would make a very fair attempt at even these if permitted. But, as I have already said before this Society, we are paying the Kafir an outrageously high wage compared with the fewness of his wants, and this point and his engagement for longer periods should be among the first to receive attention in discussing economic reforms.

Leaving education, let us turn to the health of our miners, a subject of equal importance. Thanks largely to our mine managers and the members of this Society the past 12 months has seen a betterment of underground conditions, though it is true that our managers are inclined to regard a few of our members as extremists who would like to have our underground workings used as a sanatorium and health resort by all those in need of an equable, pure and dustless atmosphere. That the efforts of certain of our members and of the Government Mines Department to secure for the Transvaal mines some degree of perfection of health conditions should have been misconstrued by some people to mean that existing conditions are bad generally is certainly unfortunate, and therefore as a participant in the recent discussion on this subject, I would again aver my personal conviction that the general health conditions attendant on mining work on the Rand are to-day as good, and in many cases better than those of any other metalliferous mining field in the world. Miners' phthisis still needs our attention, there is a lot of ground to cover before it can be run to earth, but stopped it can be for it is a preventable disease, so let us by all means see that it is prevented so far as is reasonably possible, and let us see that, where necessary, our miners are protected against themselves; but I think that we may safely rely upon Drs. Macaulay and Irvine attending to this. May I say in leaving this subject of health that had the Mines Department suggested a ·2 per cent. CO_2 vitiation limit, as was originally contemplated by Dr. Moir, I am inclined to think there would have been little opposition on the part of the mines: time would familiarise us with such a quality standard and experience would in the end have settled whether the strict standard of ·15 per cent. CO_2 was really advisable.

As a Chemical Society the subject of agriculture has on several occasions received our attention, and I believe that more than one of my predecessors has thought fit to say a few words thereon. At the present juncture, when we are hearing so much of assistance to farmers, it is perhaps as well that the agricultural potentialities of this country, the support of South Africa in the distant future, should be again brought to your notice. I speak as a child in these matters and with all deference to authorities, but a child may ask questions and I have a few I would like answered. Those who know sunny South Africa will, I think, agree with our Premier that it is indeed God's own country, but one cannot travel over it and see the hundreds, aye, thousands, of square miles untouched by the plough, unstocked by cattle or sheep, and barren of timber without receiving the impression that man has, so far, been more or less indifferent to the gift and availed himself of it but little. I fully appreciate the many troubles of our farmers here: they are such as call for indomitable pluck and perseverance, but I cannot believe, after viewing the country and after considering our exports and imports, that all has been done that might have been done. Am I wrong in saying that our farmers, as a body, are indifferent to progess, that they are content to live and are averse to the work and strife which is ever associated with advancement: I am much afraid that their past isolation and few wants have in many cases spoiled them, and that we have this, as well as numerous pests, to combat in our attempts to make the country a thriving agricultural land. Let the farmer bestir himself and do his share in helping the country along.

Turning to the imports of South Africa for the first three months of 1907 I find that we spend nearly £2,000,000 on imported articles of food and drink, this sum representing about 28 per cent. of our total imports: surely at this period of South African history this is a great reflection on our farming community. Why do we not grow more mealies? The country is well suited for such. Why do we not have more sheep and cattle farming in conjunction or otherwise with such mealie growing, why is the country not covered to a greater extent with timber for our our consumption and for export years hence when our freights are cheaper? Just imagine a country like the Transvaal with only about 500,000 cattle and 850,000 sheep on it, and worse still, I find that the total amount of land under cultivation in this colony is returned as only 1·26 per cent. of the entire area, and of this only one-half is cultivated by whites.

Mr. D. E. Hutchins, writing in 1903, said that the forests of Germany occupied one quarter of its total surface, and had a capitalised value of £900,000,000 sterling, and of Cape Colony he remarked that in 1902 the imports of timber amounted to approximately £500,000, and that for some years previous to the war the average was £250,000, and he futher mentions the fact that the Cape Government Railway which had been spending about £100,000 yearly for imported sleepers had decided to lay down special sleeper plantations, the estimated cost of which was £60,000 to £70,000, and these were expected to yield, 20 to 25 years hence, a perpetual revenue of £100,000 per annum. It is worth noting that in the year 1905-6 the mines of the Transvaal consumed timber to the value of over £600,000.

Such figures, I think you will admit, well justify the question why do we not see more timber in the Transvaal? It is because of such considerations that I think it is a matter for congratulation that the Government is giving great attention to farming, and it is to be hoped that under its fostering care the land will give a due response.

And now, gentlemen, you may be inclined to put to me that all absorbing question of the moment, how can the present acute depression, a depression which is wrecking the fortunes of so many, be removed and the future welfare of the country best assured? Remedies have been put forward from time to time, and they are again obtaining publicity before the Industrial Commissions at present sitting, and as it is largely reiteration, permit me to put them briefly. Reduce rents and bring down the cost of living; pull down freight charges, for cheap railway charges contribute largely to the development and prosperity of a country; lay out our new mines on an adequate scale and not on the kindergarten lines of the past; have large joint mills of even 1,000 stamps with centralised workshops, power station, and staffs to match; put down the minimum number of shafts; look upon labour saving devices with suspicion, for their promises are often better than their performances; afford our miners the opportunity of learning both the principles and practice of their profession; engage the Kafir on two years' contracts, and pay him a less wage for the work he does: give the miner and the Kafir each his proper sphere of work, the former using his intellectual gifts as a superintendent controlling the physical power of the coloured worker:—I submit that to have white men expending their energy on physical work which the much cheaper coloured labour is quite capable of performing is, on the face of it, absurd from a business point of view. Pay our miners a reasonable wage and let the scheme of pay be such as to encourage the men to develop and exhibit skill. See that underground supervision is made more effective, that small stopes are run by hand labour and large ones by ordinary sized machines; see that the healthiness of our working places receives due attention and insist on the miner using water jets, and plenty of water generally, to keep down dust. Workmen permanently resident in the country are desirable and, as suitable material is available, all that is necessary is its education and training. The married man has always been our best worker, so let us take care that our wage rate will suffice for his living here in comfort with his family and so induce him, if not a son of the country, to at least make it the land of his adoption. On the subject of our unskilled labour supply, the cause of such wordy strife here and such perturbation in Great Britain, I can only say that in view of the scarcity of natives practically ever since these goldfields were opened, the apparent impossibility of securing a sufficiency under the conditions hitherto obtaining, the finding of the many Commissions that have investigated this subject, the inability of the industry to forge ahead without an adequate compliment, in view of these and other reasons I consider the Chinese an unfortunate necessity, but a necessity all the same, their importation has been an expensive experiment for the mines, and it is sincerely to be hoped that no unreasoning bias against this alien labour will be allowed to cripple the mines and the revenue of the country. Cool reasoning surely counsels that we will do well to retain the Chinese until the expected substitute has materialised, and if it so happens that this great importation experiment results in converting unbelievers to the actuality of our labour scarcity and gives rises to measures which will assure our mines an adequate supply of natives on suitable terms then I think it will be admitted that the silk-clad, luxury-loving, bicycle-riding, heathen Chinese has served a good purpose, and our mining houses will probably look upon the money as not having been spent in vain.

I have here a classification of the mines of the Witwatersrand for September last, based upon their yield per ton milled. Out of the 65 mines then milling, 24 yielded not more than 30s., whilst the remaining 41 gave over 30s. Out of the total 65 mines 21 yielded between 25s. and 30s. and 20 gave between 30s. and 35s. Now these figures call for most earnest consideration. We are, as sensible business men, at the present day working the rich selected areas which offer us a profit, such areas as do not promise a due return for our capital and enterprise we naturally endeavour to avoid, though our efforts in this direction are not always successful. It may be assumed in a general way that most of the ground which offers an adequate profit under present economic conditions is being worked, and as this comprises only a small number of mines it follows that we have an immense tonnage of low grade banket on or near the outcrops as well as our great and almost intact deeper horizons of the reef, and all this reef, owing to the high cost of working, the average low grade, or the large capital which has to be risked, as yet remains largely unproved and unworked. Here I would say that I do not think that we have anything like conclusive proof of the falling away of value in depth so far as this field is concerned: in judging this matter allowance must be made for the fact that selection of stopes has been taking place, and still continues, in our existing properties. Are these low grade areas and deep zones to remain unexploited, is the country content, for instance, that only the present mines should work and working gradually fall out exhausted leaving the country to face serious financial straits only a few years hence? Surely not, and yet if this is not to be the case something of the nature of an economic revolution will have to be engineered to effect a lowering of costs, the augmenting of profits and the restoring of the confidence of the investing public. I have shown you how nearly 40 per cent. of our present mines, our selected rich areas, are not yielding more than 30s. per ton milled, and it is of interest to note that it has been recently estimated that 30s. per ton milled is probably the average value of all the unworked reef on the Rand. It is in essentially low grade rock that this country is rich, and it is the many golden miles of this that the country has to look forward to for sustenance in the years of the future. Our engineers have recently expressed the opinion that the best interests of the gold industry and the country call for a reduction of costs to about 16s. per ton milled. Now let us make every allowance for ulterior motives on their part, for even our mining engineers and financiers are not without them, let us examine their statement as critically as possible, and there will, I think, nevertheless remain a large majority of us who will agree with them that herein lies our present and future salvation, the return of prosperity, the solution of our white labour troubles, and the solid foundation upon which it will be possible to build the future of the country. Only by expansion is it possible, only by economic reform is expansion attainable, then, I say, let us have such reforms. And be it remembered that the working of very low grade ore will confer much greater benefit upon the country than upon the shareholders, the reward of the latter will be small indeed, but so long as profit is forthcoming will our gold beds be worked. Do not let our attention be distracted from the broader issues by such a comparatively small consideration, for instance, as labour ratios, this after all affects but a few of the many people in the land; do not let us, for the sake of the minority, sacrifice the majority by refusing to avail ourselves to the full of the country's rich asset of native labour, but let us earnestly and honestly assist in erecting an economic framework in the country which will stand as a fitting support for all future time. Our farming should be developed with all rapidity, our base metal and other industries should be encouraged, our present population must be provided with the wherewithal to live and settle in the country, and to do all this calls for the reforms mentioned. The country is called upon to select one of two paths—down the one there is a distant vista of a numerous and prosperous people with farming their chief vocation, substantially supported by the mining of gold, coal and base metals; down the other is seen a country undeveloped and forsaken of all but a few farmers, farmers without a market and content merely to live, a country of deserted towns and bankrupt in money and prestige, a country from which its very sons have fled to where progress holds sway. These are no idle visions but they are, I believe, faithful outline pictures of the logical outcome of the adoption of the two policies open to the country; is it possible that we will ever see the latter one? Let us hope not.

Mr. T. Lane Carter: I rise to propose a hearty vote of thanks for the able address we have heard from our new President. In listening to this address I have been reminded of a countryman of mine who started to write a history of the State of New York, and commenced that history from the time of Adam and came down the ages, and in like manner it seems to me our President to-night has commenced at the beginning and surveyed the whole field. He has touched upon many subjects in the field of mining, and there are so many points of controversy introduced into his address that I almost regret the custom in this Society of not debating the President's Inaugural Address, for I am sure many of us present would be only too pleased to get back at our worthy President. I will simply say that I appreciate his review, both of the past, present, and probable future. In regard to the uncertainty of the future and the trying term of office which will be his, I am sure we will all give him our hearty support in the difficult year he has before him. He has placed before us a picture of the future, a possible picture of a country going down to ruin, and alternatively a picture of this country as it might be if it is only given a chance. Let us hope that from all these clouds we will yet emerge, that the country will be given a chance, and that we will prove the great value of our industry for many years to come.

There is one detail he mentioned of which I should like to speak, namely, the treatment of slime in the mines in the Black Hills, U.S.A. He mentioned that slime is being successfully treated, there assaying only $\frac{3}{4}$ dwt. per ton. I have the misfortune to have slime which is not much better—just about 1 dwt. or a shade over, yet we succeed in making a considerable profit out of this low grade stuff.

Prof Yates speaks as one having authority when he deals with the question of education, and I am sure that every one of us will do his little share in helping him in his work, by recommending our miners to attend the evening classes, and in doing all we can to encourage apprentices on our mines, and wherever it is impracticable for the men to come to the classes to advise the men to enter the correspondence schools, though this instruction, of course, is inferior to the excellent lectures they get at the Transvaal University College. Prof. Yates mentioned agriculture, and said he spoke as a child on this subject. I speak not as a child, but as one who has tried it, and all that I have to say is that I wish everyone who goes in for agriculture in this country every success, but for my part I prefer to stick to mining. I have much pleasure in now proposing a hearty vote of thanks to our President for his able address.

Mr. H. A. White: In rising to second this vote of thanks I wish to draw attention to the admirable attitude adopted by our President in his address. He has not been like Achilles sulking in his tent, but like Ajax defying the lightning. Financiers, engineers and people connected with mining generally have been told recently that the effect of the hasty banishment of the Chinese would be to put them in a fit of the sulks. I think that our worthy President has given us the correct line of conduct to take, namely, that in spite of the ills forced upon us either naturally or from external sources, we will do our best to make this industry a success, and to make this Society a great part of that success.

The President: I am very much obliged to Mr. Carter and Mr. White and to all the members present for the kind way in which they have received my address, and for the promises of assistance during my tenure of office.

NOTES ON WAIHI ORE TREATMENT.

By Ralph Stokes (Member).

There are probably few gold fields in the world, from Kalgoorlie down to Bendigo, with which it may be more advantageous for the Rand metallurgist to exchange notes and experiences than the Waihi, New Zealand, whose ore has provided its exploiters with divers metallurgical problems similar to our own and whose practice has long been at a very high standard of efficiency. The Waihi Company, outside whose boundaries the field possesses little of industrial importance, employs three mills, the Victoria, Waihi and Union, of 200, 90 and 40 stamps, respectively. As at present constituted, the Waihi is the most efficient.

The object of these brief notes is not to comprehensively describe the scheme of ore reduction and treatment, but merely to draw attention to three special features likely to interest the members of this Society, namely, the honeycomb tube mill liners, the independent placing of the amalgamating house, and the vacuum slimes plant.

Primarily the characteristics of the ore and its particular requirements must be considered. The pyritic ore drawn from the lower levels of the Waihi to-day constitutes a quartz of an average hardness rarely met with in South Africa; it partakes of a chalcedonic nature which makes the necessary fine grinding a difficult operation. Before the introduction of tube mills, 1,000 lb. stamps and 40 mesh woven wire screens gave a stamp duty of 2·9 tons, and to-day, at the Waihi mill (with 20 mesh screens and 960 lb. stamps), the duty is only 3¼ tons. The increase of the mesh was naturally accompanied by the installation of regrinders. Tube mills, grinding pans and ball mills have been tried, and the first-named, on the strength of data that appeared absolutely conclusive to the management, were put into regular operation in May, 1905. At Kalgoorlie the balance of favour is with grinding pans of the improved Wheeler pattern for the fine grinding of roasted ore after Krupp ball or Griffin mills reduction and with tube mills for concentrates in connection with the Diehl bromo-cyaniding process. Thus the functions of tube mills at these two centres are dissimilar and the marked diversity of opinion is natural. At Kalgoorlie, the product is ground and reground by the flint mills until all in such a fine state of division as to be readily amenable to treatment by agitation with bromo-cyanide. At Waihi, the underflow from the spitzkasten below the stamps is passed once through the tube mills (a return is likely to be soon effected) for the production of a mixed sand and slime to be subsequently subjected to amalgamation and concentration. The relative hardness of Kalgoorlie (excluding three mines) and Waihi ores may be likened to that of semi-oxidised and pyritic ore on the Rand.

*Honeycomb Liners.**—At the Waihi mill, there are four tube mills in operation, and at the Victoria mill, five miles from the mine, nine will shortly be in use, apart from those utilised upon vanner concentrates. These mills are of the Davidsen type, 22 ft. long, and are for the most part of local manufacture. Charged with 5½ tons of flints and revolved at 27½ r.p.m., power consumption is given at 50 h.p. But the most striking feature about these mills is the honeycomb liner (Barry's patent) which strikes one immediately as the best solution of the difficulties long associated with this portion of the machines. These liners have, I believe, been introduced upon the Rand and have possibly been discussed before this Society. In forwarding these notes upon local experiences, it is therefore necessary to risk the shock of an unofficial reply anent the decease of a certain long lamented Queen of England.

For some considerable time the Waihi employed the silex blocks, but the management, bearing in mind their well-known defects and the existence of other equally durable stone, which, however, could not be fashioned into rectangular blocks, devised an admirable scheme whereby more irregular rock fragments could be economically utilised. This involved the construction of cast iron frames, 18½ in. by 15½ in. wide and 3½ in. deep, divided each way so as to comprise four compartments. The sides and dividers of the frame are about ½ in. thick on the inside, and ¼ in. at the back; so that the stones to be bedded therein are securely held. For the filling of these "honeycomb" frames a local white cherty quartz has been used with excellent results. As yet the life of these liners is uncertain, but based on six months' experience it may be calculated at nine months. As an alternative scheme, the imported flint pebbles are now being tried, but whether the greater durability more than compensates for the extra initial cost is a question yet to be determined.

The setting of the lumps or flints into the liners is a matter of as much importance as the subsequent operation of building the liners into the mill. It is essential to lasting efficiency that the stones be most carefully packed at the back of the liner, which is merely an iron plate temporarily bolted to the frame until the mortar has hardened. The condition on top, where the stones may be allowed to stand well above the frame, is of less moment than the arrangement of the rocks, in their bed of cement, so that they have the greatest possible stability. The mortar is made of Portland cement mixed with an equal proportion of ground flint fragments obtained from the tube mill. After filling, it is necessary to stack the liners until thoroughly hardened. The cement is kept wet with sacks, given an occasional hosing, and allowed to stand for a month or more. The liners, which are cast at the mine foundry, are, of course, shaped to the interior of the mill. In connection with the fitting in and repairing of the liners, whose great additional merit is that a loosening of a stone must be confined to a single compartment of 7 by 9 in., Mr. Barry, the inventor, and works superintendent of the Waihi Company, has favoured me with the following information:—

"In case any difficulty should arise in fitting the barrel segments against the rough faces of the end liners, it is more convenient to have a cast iron ring in segments cemented into the corner of the mill before starting to put in other liners. If the standard circumferential liners are a fairly neat fit, the last one in each complete ring will be slid into place, but the last one in the last ring must be made a little narrower, so that it can go straight into position, when it is keyed in with any pieces of scrap iron, wooden wedges or flat stones driven into the mortar."

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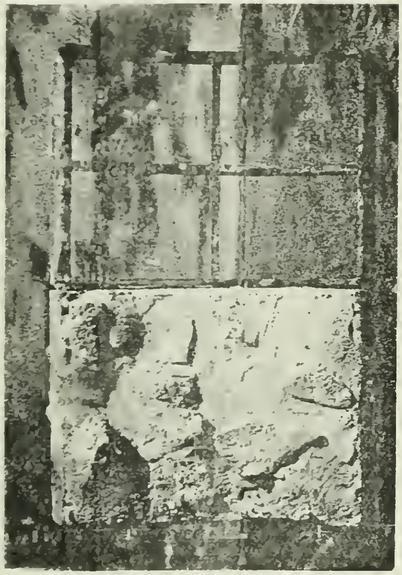
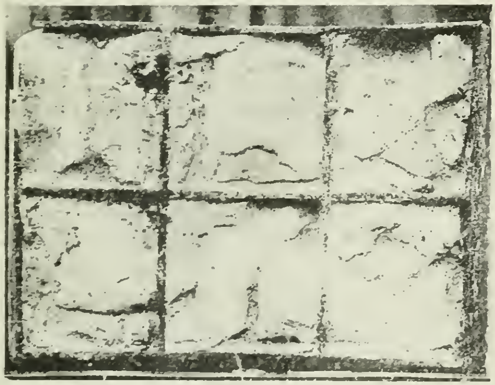
* See abstract "Grinding in Tube Mills" this *Journal*, p. 24. See also this *Journal*, vol. vii., p. 127 (Oct., 1906), p. 272 (Feb. 1907) and p. 348 (April, 1907).



HONEYCOMB LINER, EMPTY AND FULL FRAMES.



HONEYCOMB LINER AFTER SEVERAL MONTHS USE

Building Liners into Tube Mill.—The interior being clean, lay a straight row of liners, starting from one of the doors, bedding them well down in neat cement. This is a point to be carefully attended to, as some tube mill barrels are very much out of truth and a segment might bear on its edges only and have no support under the stone in the divisions. Five such rows can be laid without moving the mill. The fifth row is then to be supported by a length of wood about 3 in. by 2 in. laid along its centre. This wood is then tightly held with six stretcher bars from the opposite row of liners. The stretcher bars are simply lengths of 1 in. gas pipe with a $\frac{7}{8}$ in. bolt in one end, screwed up to a head, and an ordinary nut. The last row being thus firmly held in position so that it cannot drop down, the mill is turned round so that the last row is on top, the sixth row is then put in, the mill again shifted $\frac{1}{8}$ turn, and so on to the last or eighth row. An opening left at the closing up can be jammed tight with bits of scrap iron, old flat pieces of stone from worn out linings, or wooden wedges driven into the mortar.

Repairs.—In case the stone filling should fall out of a division it can be repaired in five minutes by putting in a spare division frame, previously filled with stone, and keying it into position with soft wood wedges."

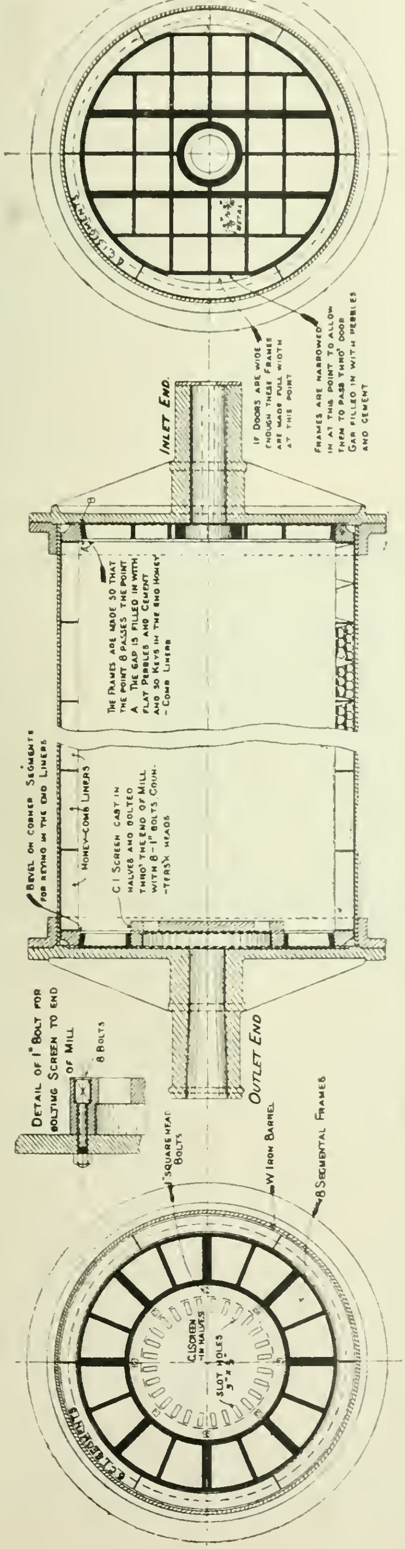
The diagram, which will be seen on the next page, further enables those interested to observe the methods employed at Waihi for the application of the liners.

Tube Mill Efficiency. — Returning to the question of tube milling generally, it must be admitted that the work done by a single transmission of sand is particularly good on such a hard material. The following is a fair parallel of sizings before and after tube milling, given to me by the chief metallurgist, Mr. Banks, a member of this Society.

	Before Tube Milling. Per cent.	After Tube Milling. Per cent.
+ 30	5·32	0·03
– 30 + 40	9·77	0·12
– 40 + 60	159·4	1·13
– 60 + 100	13·96	7·43
– 100 + 150	12·29	18·42
– 150	42·72	72·87

The cost of running is given at 1s. 2d. per ton of sand, of which flints and liners account for 7d. Each mill passes nearly 80 tons per day.

From the tube mills the general pulp flows to a large V box, from which, by means of syphon pipes, it is delivered to the amalgamating plates in a house quite apart from the stampers. There is one syphon pipe to each plate, so that the product is perfectly distributed. The advantages of this system, not to my knowledge practised anywhere else in Australasia, are several. In the first place, the full table efficiency is constantly obtained, with equal distribution, even when stamps are held up. While any plates are being cleaned, the flow from the distributing V box is returned by launder and elevator to the general inflow. The cleanly tidiness of the house—the like of which can rarely be seen outside catalogue engravings—and the prevailing quietude are also factors of importance, being conducive to careful


Detail
Inlet End.
If Doors are wide enough these Frames are made full width at this point
Frames are narrowed in at this point to allow them to pass thro' door. Gap filled in with pebbles and cement
The Frames are made so that the point B passes the point A. The gap is filled in with flat pebbles and cement and so keys in the end honey-comb liners
Bevel on corner segments for keying in the end liners
Honey-comb Liners
C.I. Screen cast in halves and bolted thro' the end of mill with 8–1" bolts. Counter-sunk heads
Detail of 1" Bolt for bolting Screen to end of Mill
8 Bolts
Outlet End
1" Square head Bolts
W. Iron Barrel
8 Segmental Frames
C.I. Screen in halves
Slot holes

PLAN SHOWING PATENT HONEYCOMB LINERS FOR ENDS OF TUBE MILLS.

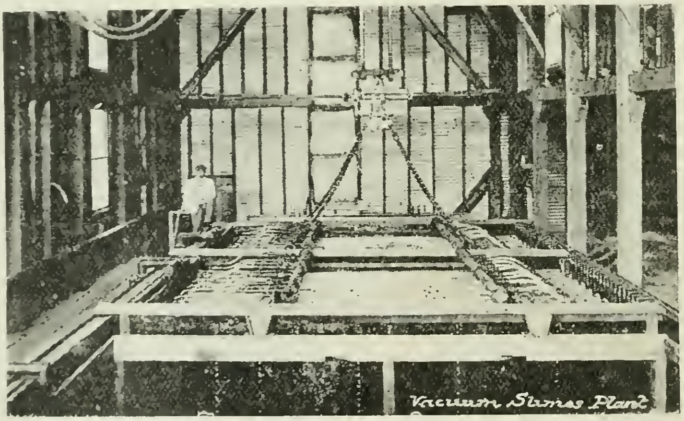
work and to a thorough understanding of queries and orders, perhaps too trivial for expression in a shout, between boss and men. Then, again, the dangers of theft are enormously reduced. Below the Waikino or Victoria 200 head mill, the amalgamating house is only entered by six men, two men per shift, excepting head amalgamator and staff officials, in the ordinary course of work.

Vacuum Filtering of Slimes.—At the Waihi mill (at the Victoria, tube mill and vacuum filter practice is not yet regularly established) the treatment of the pulp below the plates is as follows:—

Passing through a series of inverted pyramidal boxes, it is classified into coarse and fine sand and slime. The first two products pass over 30 union vanners, of the Frue type, yielding concentrates and tails; the former constituting up to 3 per cent. of the pulp, and being worth £35 to £40 per ton, the latter joining the above-mentioned slimes on their way to the main spitzkasten commanding the cyanide works. The sands are given a six-day treatment in 40 or 80 ton vats whilst the slimes pass through settling tanks, by which the water is reduced to 2 to 1, to the vacuum filter plant.

Firstly, the slime is treated in vacuum thickeners (of the same type as the filters below described) and reduced to 30 per cent. moisture, then disintegrated with a jet of cyanide solution and passed into the treatment tanks, in which it is mechanically and pneumatically agitated for 2½ days. The long cyanide treatment is necessitated by the high silver values in the ore. Waihi bullion, almost as white as tin, is only worth 12s. to 13s. per ounce, the gold being by weight about an eighth of the alloy.

After treatment the slime is run into a large rectangular tank, into which the filtering apparatus or "basket" is lowered hydraulically—though an electric elevator has been installed by the Grand Junction Company with promise of smoother operation. The vacuum filter is similar in its salient features to the Moore type, as used in the States, but differs in minor points with such results as to remove those defects observed, I understand, by a prominent member of the Society's council during an American trip some time ago. The "basket" consists of nine or eleven frames of closely corrugated iron, over which the cloth is tightly stretched. The inside vacuum pipes are applied round the frame and for an inch or two down the trough of each corrugation. Each frame is 16 ft. by 4½ ft., and the width of the "basket," across the frames, 10 ft. In the Moore machine, wooden ribbed frames are used. The vacuum, applied at 22 in., holds the slime to he cloth in a remarkably adhesive state, so that vhen raised the basket draws about 4½—5 tons f slime with 25 per cent. moisture in cakes on ach side of the frame cloth of 1¼—1½ in. thick. The filtering is effected in about 15 minutes, after vhich the laden basket is lowered into another ank of weak solution for washing for another 5 minutes, after which the waste slime residues re allowed to drop from the frame, by cutting ff the vacuum, into a hopper, whence it is sluiced way to the river. The simplicity and probable urability of these machines appear to make them reatly superior to the rotary type of vacuum lters which the writer saw at work in Kalgoorlie ix months ago, where the demand for such a nachine is even more weighty than at Waihi, or wages in Western Australia are 50 per cent. igher.



TWO BASKETS, WITH NINE FRAMES EACH, SUBMERGED. THE NEAREST VAT (EMPTY) IS USED FOR WASHING SLIME WITH WEAK CYANIDE SOLUTION.

By comparing costs and efficiencies of the resses and vacuum filters, the Waihi has already iad sufficient experience of the new installation o learn that there are even no grounds for controversy on the matter The presses will soon be ompletely ousted. At the Waihi mill, where he filters have been used for several months, 220 ons of slime are put through per day by means f two baskets. The installation cost about £4,000. Labour requirements are three men, ne per shift, and the part services of a fitter.

Running costs are :—

	Per day.
Three men at 8s.	24s.
Cloths at ½d. per ton treated	9s. 2d.
Part services of fitter ...	3s.
Two vacuum pumps (17 h.p.)	30s.
Power for pneumatic agitation	10s.
	£3 16s. 2d.

Or, say, 4½d. per ton treated. At the Waihi, this means a saving of 1s. 3d. per ton of slime as compared with the Johnson presses.

So conclusively successful has the coincident introduction of tube mills and vacuum filters proved upon the property—and it is necessary to bear in mind that the product of the tube mill as at present dealt with includes a quantity of fine sand which passes the spitz and finds its way to the slimes plant—so favourable have been the results of lengthy experimenting and practical work, that the controlling heads are already beginning to speak in an undertone of the practicable extension to a far greater scope of fine grinding and filtering; in fact, to use the magic term, "all sliming." The question is beyond the limits of this patchy contribution; but it is one which cannot well be judged by Rand standards.

In conclusion, it may be noted that the rich concentrates won at each mill, by vanners at the Waihi mill from fine and coarse sand, and by Wilfley tables at the Victoria from only the heavier sand, are all collected and treated at a plant in connection with the latter works. After being carefully weighed and sampled the concentrates are passed into tube mills, whose product is elevated and classified by a spitzkasten, returning the coarse to the tubes, whilst the fine is thickened and treated by cyanide solution made up to ·03 strength for about seven days in air agitation tanks 14 ft. deep and 5½ ft. in diameter. An extraction of 95 per cent. gold and silver is obtained.

As for the total extraction, official records, which are kept with the most careful scrutiny, show an improving average of 90 per cent. gold and 75 per cent. silver. The residues, which run away to the river, have already proved the snare and ruin of one adventurous syndicate.

The President: We have been listening to a paper which has come to us from the South Pacific, from one of the earth's beauty spots, Fiji; it might almost be said that it was written under the shade of the palms. Mr. Stokes, the author, is travelling round the world, visiting the various mining centres, and, I think, it is very kind of him to remember the Society when he is so far away from us. The Waihi Mine, New Zealand, is one of the best of the world's gold mines, and one of the largest. Its monthly output is about £64,000 sterling from a tonnage of about 27,000, so that it is also a high grade proposition. I move that a special vote of thanks be sent to Mr. Stokes for his paper.

This was agreed to.

LAST DRAININGS.

(*Read at February Meeting, 1907.*)

By H. A. White (Member).

REPLY TO DISCUSSION.

Mr. H. A. White: I have to thank those gentlemen who have contributed to the discussion on this paper and to congratulate this Society on two able recruits to the ranks of active participators in our proceedings. It is pleasing to find that the general principles set forth have been fully endorsed and found to be useful in practice. I believe that occasional papers of this kind, which endeavour to deal with the principles underlying the best modern cyanide treatment, will always be acceptable and make up in usefulness what they must lack in novelty. I fully agree with Mr. Dowling, Mr. Pyles and Mr. Lea that draining dry after each saturation will give the best results where sufficient time is available. This method uses each ton of solution to the best advantage but, of course, if the plant provided is not large enough experiments must be made to determine the best treatment possible. On this point I have found the difference between the two methods to be less than would be anticipated on a casual consideration of the case. I find in strictly comparative experiments on a working scale that 176 tons of solution applied during leaching dry eight times is almost exactly equal in effect, as far as concerns last drainings, to 184 tons applied during five leachings dry. On a circumscribed plant it is possible that the one method might be used in winter when sumps are usually a shade higher than in summer and the relief of slightly less tonnage would be felt.

I would point out to Mr. Dowling that though slimes solutions are precipitated most satisfactorily it must not be overlooked that the zinc in these boxes is mostly newly prepared and is at a great advantage when compared with that in weak and medium boxes. I think in the warm weather a shade less than 0·1 per cent. KCy might be profitable, but my experience shows no advantage in reducing this strength in winter. It is not the solution but the precipitation of the gold which is in this case the deciding factor.

I do not quarrel with Mr. Lea's figure of 2d. per ton or ·04 dwt. as a cost of increasing solution pumped and precipitated. My object in taking 6d. as a working hypothesis was to show profits to be derived even on small or poorly equipped plants by a careful examination of results obtained by varying methods.

I notice in the daily papers that one small group of mines in an outlying district found trouble over trying to economise cyanide and reports more gold left behind in the boxes after clean up than usual. This might have been avoided by classing the solutions in accordance with gold values and ignoring cyanide strength. The method Mr. Dowling illustrated in his contribution is similar to that I recommend, save the simplification of opening at once to strong boxes, which I find reduces white precipitate and improves results generally as the following figures for three months show:—

Strong (rich)	·57 per cent. KCy,	value 4·57 dwt.
Medium	·41 per cent. KCy,	value 0·82 dwt.
Weak (poor)	·33 per cent. KCy,	value 0·42 dwt.

Mr. Fraser Alexander feeds fat his ancient grudge against the mining department in the matter of bunches of acid ore, and all will agree with him that this point deserves, and will eventually obtain, especial treatment. Not only the sands plant but the slimes plant also show the bad effect of dumping, without warning or any precaution, a few tons now and again of acid or otherwise contaminated material into the stream of ore from which the reduction works has to charm away the valuable metals.

I hope Mr. McNaughtan will not deem it impertinent to suggest that our monthly meetings, even in these depressed times, are worth coming in for. If his promises are fulfilled I will guarantee that eight or nine leachings dry will soon cease to mystify.

I have to express regret that no one has supported my suggestion to register cyanide strength and alkalinity in terms of the effective ions. Possibly we are somewhat shy of dealing with these newer developments of chemical theory in spite of their able exposition by Prof. Oettingen before this Society eight years ago.*

It is to be remarked, in passing, that to speak of loss of cyanide in the boxes is only a *façon de parler*, as total cyanogen is practically the same at the head and the foot, and that it is in the tanks and treatment vats where the real loss takes place. As our records are in terms of available cyanide, this custom has, of course, some little justification beyond that of mere convenience.

Mr. Pyles's contribution to the method of dealing with lumps of slime is very welcome, and, though he does not report complete success, he has given us some useful particulars of his results. He agrees with Mr. Lea that a tube mill product shows no necessity for including any "standing dry" in the treatment. All the figures brought forward go to show that the bulk of the gold is dissolved very quickly, but as Mr. Dowling points out, a small amount goes into solution almost, if not quite, to the end of the treatment. This explains the possibility of Mr. Lea's result showing ·11 dwt. increase in value of drainings after transfer to second treatment. Mr. Lea can be congratulated on the adequacy of his plant which reflects credit on this Society, since one of our past presidents designed it. His very low results do him great credit, and it is in no envious spirit that I point out a possible drawback. I refer to the results shown after transfer to second treatment vats. 290 tons of solution are obtained of an average value ·09 dwt. Not all of this gold is derived from the sand, a "trace" in the sumps may be more than ·01 dwt. Again a part of this gold must go to increase value of slimes residues if some of the solution is used for second transfer there.

Assume, however, that 25 dwt. of gold are actually recovered from the sand and put against that the cost of extra handling, maintenance, interest, and amortisation on the second treatment tanks and cost of elevating to the additional height, and you will find that 360 tons of sand and 290 tons of solution will scarcely repay the cost of bringing them together. If, however, his figures show a margin of profit on this apparently discouraging proposition, however small, I entirely agree that the policy adopted is sound commercial economy. I am pleased to notice Mr. Lea's reference to the direct application of strong solution by Mr. Dowling as early as 1904, as it is by no means surprising to us, who know him, to find him well abreast of the times, and indeed it is my duty and pleasure to thank him and others of our members for useful information, the ungrudging imparting of which is part of the great reputation of our Society.

MINE SUBSIDENCE.

(Read at March Meeting, 1907.)

By Alex. Richardson, A.I.M.M. (Member).

DISCUSSION.

Mr. H. Weldon: Mr. Richardson has dealt with a subject of peculiar interest, and it is satisfactory to know that his investigations have demonstrated the value of the recognised formula in regard to subsidences due to the collapse of mine excavations.

In regard to the support of roof in underground workings, there can be no doubt that as Mr. Richardson points out—"careful and efficient stowing" is the most reliable method. In our mines, in the case of steeply-inclined underhand stopes, there can be no question that stulls loaded with waste rock form a very safe, cheap and adequate support, whilst in overhand stopes, nothing could be better than to allow portion of the broken reef to remain in and partially fill the stope. Sorting in the mine has been suggested and tried from time to time, but the practice has not so far been taken up. I would suggest, however, that the difficulty experienced in some mines of obtaining sufficient waste rock to pack the stulls with, could be overcome by shooting down portion of the footwall by means of a series of short holes.

I note that Mr. Richardson estimates the ultimate strength of mine pillars in regard to their power to resist crushing, as 10,000 lb. per sq. in., and that these conclusions are based upon results of tests carried out on 4 in. cubes of local quartzite. Such a constant, however, only holds true when the height and sides of the pillar are of equal dimensions, but this is not usually the case in mine pillars, the length and breadth of which generally bear a proportion to their height of 2 or 3 to 1. Now we know that the power of a pillar (according to the formula laid down by Euler and Eytelwein) to resist crushing increases according to the square of its area, so that an ordinary mine pillar, the sides of which measure 10 ft., in a stope of 5 ft. in width, would have a strength of not less than 16 times that of a pillar of the same height with sides measuring 5 ft. In view of the rapid increase in weight-bearing capacity of a pillar for any given increase in its superficial area, the somewhat alarming conclusions arrived at in the percentage table given in Mr. Richardson's paper must be regarded as unwarranted and not borne out by practical experience.

In regard to the next table setting forth the bending and shearing factors, the conclusions arrived at are certainly not in accordance with mining experience in these mines, which is to the effect that shearing or punching of the roof is an extremely rare occurrence. This discrepancy between the theoretical and actual results may be due to the fact that the hanging wall is never of a homogeneous nature, lines of weakness being always present, and therefore vitiating any theoretical calculations.

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* See Proceedings, vol. ii., pp. 543 and 556.

The size of shaft pillars, the distance between stope pillars, and the diameters of the same are factors of the greatest importance, and it would be of great interest to learn the details on which these calculations are based.

I think that on these fields where we have such large underground areas on which we have buildings to protect, that this paper of Mr. Richardson's is particularly interesting. I am sure there are many more points which I have passed over, but these in particular struck me when I read the paper.

The President: I am sure we have to thank Mr. Weldon for coming forward and giving us his views. He has certainly raised more than one interesting point, and we wish he would attend our meetings more frequently. I would like to take this opportunity to say how indebted we are to him for the excellent report issued by the Mines Department. In this matter of mine subsidence, for instance, I can assure you that you will find the diagrams, etc., in this report most instructive, and I think it is only just to the Mines Department that this should be known.

Mr. E. Homersham: I cannot agree with Mr. Weldon that it would be a safe proceeding on the Witwatersrand generally to break into the footwall to get mullock for loading up stulls, as it would make the stope dangerous on account of the liability of slabs of footwall coming away from the slipping faces that usually exist in foot and hanging walls all through our mines, certainly at Randfontein it would be an impossible procedure.

In answer to the Chairman Mr. Homersham said the reefs at Randfontein had a steep dip.

Mr. M. H. Coombe: We do not, on the Rand, adopt the practice which is followed in most countries to ensure safe mining. For instance we do but little "resuing." In stopes of more than 50 deg. dip this mode of stoping would be more economical in that we would get cleaner reef for the mill, and falls of ground from foot and hanging would be reduced to a minimum. Mr. Richardson's paper is an admirable one but whilst listening to the discussion here to-night I have been wondering, if it is possible for a number of gentlemen to gather together, and from formulæ of the crushing strength of various rocks, and the angles of cleavages, etc., lay down the law for the leaving of pillars, both in regard to position and size, for the support of the walls of a mine. With all respect to their learning and theory I say they cannot do it; this sort of thing comes under the ægis of the practical man, be he workman, shift boss, or mine foreman. And here I submit a grievance that undoubtedly exists amongst our older and most experienced miners. In a good many cases when a vacancy occurs on the underground staff instead of one of the best workmen being promoted, a young beardless lad from a School of Mines is put into the position and given authority over men with 20, 30 and 40 years experience of mining in all parts of the world. Educate young men you must in the higher branches of the profession, but when they have finished their education and graduated don't give them charge of miners and thus create inefficiency and danger underground but first give them three or four years of practical mining work and thus turn out men who may become, if it lies in them, as near a perfect mine manager as can be created. There are plenty of men to draw from on our mines, with large experience, and if there were more of this type in charge of underground work here less would be heard of falls of ground and consequent loss of assets, or of life and limb.

Mr. S. H. Boright (contributed): I send a section of the strata taken at a point 400 ft. from the subsidence at the Great Eastern Collieries. This section evidently does not hold good at the point of subsidence as will be seen later.

SECTION OF STRATA PASSED THROUGH IN SINKING "OLD MAIN SHAFT," GREAT EASTERN COLLIERIES.

Formation.	Thickness in feet.	Depth from surface to bottom of formation in feet.
Surface soil and clay -	23	23
Soft sandstone - -	64	87
Shale - - - -	8	95
Sandstone - - -	56	151
Inferior coal and shale -	65	216
Top coal - - -	11	227
Good coal - - -	19	245
Inferior coal - -	7	252
Chert - - - -	—	—

Our bords were worked from 15 to 20 ft. wide, leaving pillars 30 ft. sq. During the 10 years that these workings have been standing the pillars had reduced themselves by flaking to about 25 ft. sq., thus enlarging the bords by a like width, but the pillars were all in good condition, solid throughout, without any cracks or slips. The workings at this part of the colliery were from 15 ft. to 25 ft. high and at the exact point of subsidence they were about 20 ft., I think—possibly 25 ft. About 60 ft. from the subsidence, and on all sides of it, there were either pillars 60 ft. x 60 ft. or else solid ground. On the East side of the subsidence is a large dyke about 60 to 80 ft. from the edge of the subsidence. The coal had been mined up to the dyke but was solid beyond the latter. 60 ft. South is our boundary pillar and the same distance West are two rows of pillars 60 ft. x 60 ft. and the same on the North.

There had been, for some months before this subsidence occurred, a great many minor falls of roof both at this pit and at our other pit four miles away, and I think it will be found that falls were more prevalent throughout the Rand before the month of May than they have been since. There was no warning at this particular point before the fall but we had heard falls on the adjoining property beyond the boundary pillar, at a point some distance West of where the subsidence took place. The latter is roughly circular and 250 ft. in diameter; the depth is about 40 ft., and the sides nearly vertical. The underground dimensions are about 50 ft. larger than on the surface. The ground exposed by the fall is all surface soil and clay. Underground the fallen material consists of coal, shale, decomposed sandstone, and surface material. It seems that at this particular point there was very little sandstone and that the influence of the dyke had decomposed what there was. The pillars seem to have stood under the fall, and the roof of the bords given way under the direct pressure of the loose strata. I am firmly convinced that there would have been no fall if the sandstone above had been of the same quality as that shown in the section a short distance away.

NOTES ON SOME RECENT IMPROVEMENTS IN TUBE MILL PRACTICE.

(Read at April Meeting, 1907.)

By Kenneth L. Graham, M.I.M.M. (Member).

DISCUSSION.

Mr. Jas. Lea: Although banket has been used in tube mills for so long, as a substitute for pebbles, the comparative test of pebbles versus mine ore, made by Mr. Graham, and published in his paper, is the first definite information we have had on the subject. Since it appears to be now generally accepted amongst tube mill men, that banket is quite as efficient as pebbles as a grinding agent, I should like to draw the attention of members to the size of the banket fed in tube mills. There appears to be a disposition on the part of some, to emulate the stamp battery as a crushing machine, which I venture to say, is developing in the wrong direction. The smaller the pieces of ore fed into the mill, the sooner they become inefficient as pebbles, and the greater the total amount required to be fed. As banket is consumed at such an enormous rate compared to pebbles, and, as by the time the corners are worn off banket, to make it a decent shaped pebble, it is probably reduced from 4½ in. to 3 in., it seems to me that we should start off with as large a piece of ore as possible, consistent with the object in view; that is, to have an efficient pebble by the time it has been knocked into shape.

I have not the convenience of an automatic feed, such as described by Mr. Graham, which must so greatly assist where the consumption is so high, but I find, being restricted to hand feeding, the latter has one very great advantage over an automatic feed with the present diameter of feed ends of mills, namely, that a larger pebble can be fed with less danger of choking. I have no difficulty in maintaining the pebble load by hand.

No. 1 Fraser & Chalmers mill, 22 ft. x 5 ft., has a 4½ in. feed aperture, and by feeding in ore of slightly less size and keeping the load of pebbles up to throttling point—that is, well over half full—I find by actual weighing 3 tons of ore per day as much as the mill will consume. This is when using a 1⅜ in. nozzle on the dewaterer, and the pulp containing an average of 37 per cent. moisture.

With a feed carrying 37 per cent. moisture I find that, if the pebble load exceeds half full by much more than about 6 in. back flow takes place.

On the old feed end of a No. 2 Allis-Chalmers mill, 22 ft. x 5 ft., wearing through, a new one was obtained of a slightly different design. The old hopper was retained, but the back of the feed end, instead of being solid was left open, and a bend cast on. By this means independent pulp and pebble feed apertures were obtained, and the pebble feed aperture was enlarged to 6 in. thereby taking a pebble as large as the feed end liner would allow. I find this entails less labour to maintain the load, and the consumption in this mill is considerably less than 3 tons per day.

The use of mine ore for grinding seems to me to demand a slight modification in the design of the feed end, and I would suggest that the diameter be increased, so as to take a 7 in. or

8 in. piece of ore. By this means we should obtain a correspondingly increased life to the pebble, and I believe the results would be equally satisfactory, and the much laboured mine ore consumption would be immensely reduced.

While on the subject of pebbles, I think the discharge screen requires a little more attention paid to it, than has been the case so far. Present practice seems to be to wear the pebble as near to pulp as possible. There must obviously be a limit to the size of an efficient pebble if we are to keep the tube mills for regrinding battery product. It is customary, I believe, to use a discharge screen having apertures of about ¼ in., but, if a pebble is retained in the mill until it is the size of a pea, it is certainly not going to crush anything by impact, and I should imagine it will have lost its virtue as a grinder, and, therefore, should not be in the mill, where it is occupying the space that should be taken up by an efficient pebble. I believe the Australian practice is to periodically sort out the small pebbles; this, however, can be regulated by the discharge screen apertures being larger.

The size of nozzle for the dewaterer is governed by a multitude of conditions, or lack of them, not the least being the number of mills installed per 100 stamps, and the degree of fineness to which it is desirable to grind. With a sufficient number of mills it may probably be advisable to use a smaller nozzle than on a plant where there are not enough. But as we are equipped insufficiently on these fields the greater the quantity of pulp, properly classified, passed through a mill, the greater the quantity that will be reground, and the smaller the percentage + ·01 grade entering the cyanide works. I find 1½ in. the most efficient size nozzle to use from a regrinding point of view, giving 390 tons of dry sand per day entering at an average moisture of 37 per cent. I would point out that the nozzle used in our dewaterer is fixed in the bottom of the spitzkasten itself, and is 2 in. long, this being the total distance over which there is any pipe friction. These nozzles do not wear at the top for some days, but gradually wear from the bottom upwards, eventually leaving just a ring at the top of the original diameter, but the difference in the friction of a parallel nozzle 2 in. long and one that has worn slightly from the bottom up makes a very considerable difference in the tonnage passed. I mention this by way of emphasising the pipe friction spoken of by Mr. Alexander.

The number of shaking tables should bear some relation to the quantity of pulp passing through the tube. Four tables seem to be the regulation number per tube irrespective of tonnage, but, I think, where large quantities are passed through the mill six tables would be safer and would allow for overloading when scraping, dressing, repairing, etc.

The quick setting cement mentioned by Mr. Graham should go a long way towards reducing the time tube mills are hung up for relining; but, it is for the individual tube mill or cyanide manager to ascertain, whether it is more economical to use this very expensive cement or hang up the mill for an extra day. For instance, this quick setting cement costs, I believe, £65 per mill, as against £10 for ordinary cement, a difference of £55. Now, take a battery producing 800 tons of sand a day, this £55 would allow for an increase in the residue value of ·344 dwt. per ton, more, of course, with a smaller tonnage, and I think this is a larger increase than most cyanide managers find to be the case with one tube mill hung up, thereby not justifying the expense. I am not in any way arguing against reducing the time for relining, as I think there is room for a considerable improvement, but it should be done with a view to the greatest profit, not the shortest time irrespective of cost.

Of the last two mills relined on the Robinson Deep, No. 1 was stopped and handed over to the engineer at 6 a.m., it was then discharged, relined with local chert and ordinary cement, reloaded and under steam at 7 a.m. the next morning, and after 24 hours under steam was started up at 7 a.m. the second day, the total stoppage being 49 hours. No. 2 mill was stopped at 6 a.m., relined with silex and ordinary cement, reloaded and under steam at 8 a.m. the next morning, and after 24 hours under steam was started at 8 a.m. the second day, the total stoppage being 50 hours. On inspection, both mills are wearing well and do not show signs of weakness. I am of the opinion that the time for relining can still be considerably reduced, yet, if the quick setting cement used by Mr. Graham had been used in these mills, both would have been running inside 29 hours, if, as I believe, Mr. Graham allows three hours to set.

With reference to the increase of temperature of pulp passing through the mills, the highest I have been able to record is 5 deg. C.

The President: I think it would be as well to keep this discussion open until the next meeting. It is very interesting, and I can well understand that there are many more members who would like to come forward to discuss it.

This was agreed to.

A FEW NOTES ON THE REFINING OF BASE BULLION.

(*Read at May Meeting, 1907.*)

By C. W. Lee and W. O. Brunton (Members).

DISCUSSION.

Dr. T. K. Rose: The paper is interesting as another illustration of the care taken in South Africa not to let slip anything which may be of value in the mining industry. The method of procedure adopted by Messrs. Lee and Brunton, seems ingenious and satisfactory. I should be glad to know whether the clay tubes were found to wear away rapidly. With Morgan's thick-walled tubes, 1 in. in diameter, with $\frac{1}{8}$ in. bore, we have found the wear to be insignificant in toughening brittle standard gold at the Royal Mint, but the work lasts only a few minutes in this case, as the impurity to be removed, usually lead, is always less than 1 part per 1,000. The cost is hardly more than that of ordinary melting. In these cases we pour the slag with the metal and scrape off the still liquid or viscous slag as soon as the metal has solidified. I have not tried skimming off the slag in the pot, but chipping off the solid boro-silicates is certainly a very tedious operation.

The meeting then closed.

Errata.

Table of Melting Points.

(See this *Journal*, March, 1907, p. 297.)

Since the above table has been published several errata have been pointed out, namely:—

The temperatures for O and F are below zero, and, therefore, should have the sign minus in front.

The melting point of Mg is 633 deg., not 800 (Heycock and Neville, 1895).

The latest investigation of Holborn and Valentiner (*Annalen der Physik*, vol. 22, 1907, pp. 1—48) gives the melting point of Pt 1,790 deg. C. and Pd 1,575 deg. C.

The melting point of Ni is 1,484 deg. C. and Sn, 241 deg. C. (earlier observers placing this at 232 deg. C.).

The melting point of Ca is 800 deg. C., not 322.

In column viii. the metals should read Ru, Rh, Pd, and not as printed.—(*Electrochemical and Metallurgical Industry*, March and May, 1907.) (J. A. W.)

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Determination of Manganese in Tungsten Steels.—"Tungsten interferes with the determination of manganese by the author's persulphate method; so does molybdenum to a less serious extent. If the amount of peroxide precipitated from the boiling solution of persulphate is determined volumetrically by hydrogen peroxide, the error caused by tungsten is much more serious than when ferrous sulphate is used for the determination; but even in the latter case it is enough entirely to vitiate the results. In the case of tungsten steels, the error is quite avoided by dissolving the sample in the absence of air, when the tungsten remains undissolved and can be filtered off, the small amount oxidised and dissolved during the filtration and washing, being insufficient to affect the determination of manganese. A sufficiency of dilute sulphuric acid is placed in a capacious conical flask, saturated solution of sodium bicarbonate added till the flask is filled with carbon dioxide, and the weighed sample (2—10 gm., according to the amount of manganese) introduced. The flask is then closed by a cork which carries a twice-bent tube, drawn out to a point at the inner end just below the cork, and dipping at the other end under saturated sodium bicarbonate solution. The liquid is warmed till solution is complete, and finally boiled for a few minutes. After cooling, the undissolved tungsten is filtered off and washed two or three times, and the filtrate is treated with 10—12 per cent. ammonium persulphate in small quantities till the iron is peroxidised, then 30—40 c c. are added in excess, the liquid diluted to 400—500 c.c., boiled for 20 minutes, and the precipitated manganese peroxide filtered off and washed, dissolved in a known volume of acidified ferrous sulphate solution of known titre, and the unaltered ferrous salt determined."—G. v. Knorre. — *Journal of the Society of Chemical Industry*, April 15, 1907, p. 345. (A. W.)

Acetylene as a Precipitant in Analysis.—"Copper is easily and completely precipitated from cuprous solutions, either ammoniacal or acidified with acetic or tartaric acid, by acetylene, as a granular, easily filterable precipitate of cuprous acetylide, $C_2H_2Cu_2O$. The authors use an amount of substance containing 0·1 to 0·3 gm. of copper, dissolved in 100 c.c. of water. To this is added 10 c c. of ammonia of sp. gr. 0·96, and 5 c c. of a 10 per cent. solution of hydroxylamine hydrochloride to reduce to the cuprous condition, and the precipitation is effected either by a current of purified acetylene, or by adding a solution of acetylene in acetone, about 5 c.c. of which is sufficient. The precipitate is filtered off and washed with warm and then cold water to which a little acetylene water has been added; the filter and moist precipitate are then placed in a porcelain crucible of about 75 c.c. capacity, treated with 10—15 c.c. of nitric acid of sp. gr. 1·15, and heated on the water-bath. When decomposition is complete, a few drops of acid, of sp. gr. 1·52, are added, and the whole evaporated to dryness, then gradually heated to ignition, preferably with addition of a little powdered oxalic acid, to reduce any nitrocellulose which might otherwise decompose explosively, and the cupric oxide weighed. Of the other common metals, silver and mercury are the only ones precipitated by acetylene, so that they must be first removed if present, the silver by precipitation as chloride, the mercury by reduction in hydrochloric acid solution to calomel, by means of phosphorous acid. The phosphorous acid does not interfere with the precipitation of the copper. The separation of copper from the alkaline-earth metals, magnesium or cadmium, must be carried out in acetic or tartaric acid solution, to prevent the formation of carbonates, or the precipitate, if formed in ammoniacal solution, must be washed with weak acetic or tartaric acid. If aluminium, chromium, manganese, iron, or cobalt be present, tartaric acid must be added in sufficient amount to prevent precipitation when the solution is made ammoniacal. The same precaution applies to bismuth, antimony, and tin. The separation from lead is best effected in acetic acid solution. All of these separations are sharp and complete. Palladium is precipitated completely by acetylene from solutions acidified with hydrochloric acid. The precipitate is easily filtered and washed, and is quietly decomposed by heat, giving an oxygen compound of palladium, which is easily reduced to metal in a stream of hydrogen. In the filtrate, after rendering it ammoniacal and reducing by hydroxylamine, copper can be precipitated by acetylene. The acetylene method allows of separating palladium directly from platinum and its congeners, except osmium. This metal is precipitated by acetylene as metallic osmium; the mixed precipitate, when washed, dried, and ignited in hydrogen, is weighed as mixed metals, and the osmium is then volatilised by heating in a stream of oxygen, and the remaining palladium again weighed. Gold is also precipitated as metal by acetylene: and if it is to be separated from palladium, the mixed metals obtained, as in the case of osmium, must first be weighed, and then the palladium dissolved out by nitric acid."—O. MAKOWKA, Z. anal. Chem., 1907. *46*, 125—128; 128—141; 141—145; 145—150.—*Journal of the Society of Chemical Industry*, April 15, 1907, pp. 344-5. (A. W.)

STANDARDISING ACIDS WITH MAGNESIUM.—"The author has used metallic magnesium to standardise acids for volumetric work. Having made approximate solutions of acid and alkali, and ascertained their value in terms of one another, he dissolves in a measured volume of the acid an accurately weighed quantity of magnesium ribbon (equivalent to not more than nine-tenths of the acid taken, so as to ensure complete solution within a reasonable time), and then titrates the excess of acid with the alkali solution."—A. VESTERBERG, Z. anal. Chem., 1907, *47*, 81—83.—*Journal of the Society of Chemical Industry*, April 15, 1907, p. 344. (A. W.)

FORMATION OF CALCIUM CYANAMIDE.—"The fixation of nitrogen by calcium carbide, according to the process of Frank and Caro, with the formation of calcium cyanamide, has been studied, particular attention being paid to the influence upon the reaction of temperature and admixture of calcium chloride or fluoride. Below 1,000° C., the absorption of nitrogen by finely powdered calcium carbide is limited and takes place very slowly, but between 1,050° C. and 1,100° C. the reaction readily occurs and proceeds rapidly, until the theoretical quantity of nitrogen required to form calcium cyanamide CaN_2C, has been fixed. Admixture of calcium chloride, as first observed by Polzenius (Ger. Pat. 163,320), lowers the temperature of reaction to a very marked extent, a rapid absorption occurring from 700° C. upwards. The influence of the proportion of calcium chloride has been studied, it being found that the rate of absorption increases with the content of calcium chloride up to 30 per cent., although the effect becomes less marked as the percentage increases. Admixture of calcium fluoride with the carbide is also advantageous, but the temperature of reaction is not brought quite so low as in the case of the chloride. Here an addition of 2 to 5 per cent. seems to be most favourable, since with an increase of percentage of the fluoride the quantity of nitrogen absorbed in a given time diminishes. In explanation of the effect of admixtures of salts the authors suggest that the removal of the film of cyanamide from the carbide may assist the reaction; the softening and fusion of the mass may also increase its power of dissolving nitrogen. At 800° C. in two hours, mixtures of calcium carbide with 15 per cent. of calcium chloride reached a content of 17·6 per cent. of nitrogen, with the gas under a water pressure of 132 mm., and 19·9 per cent. with a pressure of 1,840 mm., thus showing a slight favourable influence of pressure."—F. FOERSTER and H. JACOBY, Z. Elektrochem, 1907, *13*, 101—107.—*Journal of the Society of Chemical Industry*, April 30, 1907, p. 423. (A. W.)

NEW METHOD OF DETERMINING AMMONIA IN WATER.—"The method proposed is based on the precipitation of the ammonia by mercuric chloride in the presence of sodium carbonate, and permits of the determination of quantities of ammonia varying from 0·0001 to 0·012 gm. per litre. The details of the method are:—One litre of the water to be examined is rendered alkaline by the addition of sodium hydroxide and distilled slowly until about 100 c.c. of distillate have been collected, the latter being received in a vessel containing 10 c.c. of 1 per cent. hydrochloric acid. If much ammonia is present, the distillate is diluted, but not in cases where the ammonia is less than 0·001 gm. per litre. Ten c.c. of 5 per cent. mercuric chloride solution and 10 c.c. of 15 per cent. sodium carbonate solution are added to the distillate, the mixture is placed aside for 24 hours and the precipitate then collected on a weighed filter, washed with 5 c.c. of water, dried at a temperature of 100° C., and weighed. The weight of the precipitate is multiplied by 0·03 to give the amount of ammonia in 1 litre of the water. The composition of the precipitate is only constant when less than 0·5 gm. of mercuric chloride per litre is employed for the precipitation."—A. BUISSON. Compt. Rend., 1907, *144*, 493—495.—*Journal of the Society of Chemical Industry*, April 15, 1907, p. 346. (A. W.)

MANUFACTURE OF EXPLOSIVES.—"In the course of the discussion on the vote for war stores in the House of Commons recently, Mr. Haldane, speaking of the tests applied to cordite, said that no cordite about which there was any suspicion would be made available for use until it had passed the test. The official reticence about the Woolwich explosion had nothing whatever to do with mercuric chloride. The Government had there samples of certain explosives which it was thought might put us ahead of all other nations; but these samples had now, unfortunately, disappeared into space. In black powder the particles of carbon, and the atoms of oxygen which were in combination with something else, lay side by side. These two substances side by side were very stable, and when the flash ignited the powder the oxygen was freed and had to travel a certain space, very small, and unite with the carbon, and they had a slow explosion and a great deal of smoke. Nitro-glycerine and gun-cotton, which were the ingredients of cordite, had wholly different properties. In gun-cotton and nitro-glycerine the atoms of oxygen and carbon lay together in one very complex molecule. The atoms of oxygen, when released, rushed very quickly to the carbon and produced a tremendous explosion. The result was that all these explosives were by nature dangerous. It was quite true that black powder, on the other hand, was a safe explosive, but black powder was feeble and emitted smoke, and any nation which used it would be 'out of it' with other nations. They had, therefore, to deal with chemical problems in order to do what they could to tame the dangerous explosives. The discovery of the means by which to tame these explosives was a very remarkable one. The nitro-glycerine and the gun-cotton had been found to be capable of being rolled together under heat, and mixed together with a solvent, and so incorporated that they made a kind of paste which would roll into a gelatinised form and assume a character which made them safe to handle and, which was much more important, safe to burn and capable of burning slowly. The process of kneading them together and making them into a horny substance made them perfectly safe, unless they changed their character, and it gave them a quality which remained for some time unchanged. All powders of whatever description in which the carbon and oxygen were in the same molecule were prone to deteriorate, particularly under heat. To some extent also the nitrogen compounds might be given off, and might assume by contact with the atmosphere the shape of nitrous acid or nitric acid. In that way reactions might be caused in the powder, making bad spots and developing heat which might cause local explosions that were communicated to the whole mass. The result was that they had to test in the very closest fashion to see whether the cordite was properly made, and not only so, but they had to test after certain intervals to see whether it had not deteriorated. He said, therefore, that the very best cordite made by the best process could not be trusted if exposed for a prolonged period to an unduly high temperature. That was why they were so careful about the magazines of ships, because in a hot climate like India there was danger, and also in an unduly cold climate, where the nitro-glycerine was frozen out. All cordite, therefore, was subjected to the heat test, to set free and disintegrate the nitrogen compounds of which he had spoken. Very elaborate and delicate tests were made by means of fine tissue paper steeped in starch, which was so exposed at a given temperature to the action of the fumes from the cordite that it would show certain lines, according as the fumes were given off abnormally quickly or normally, just as they should under the high temperature at which the test was conducted. Now, mercuric chloride had the property of preventing the giving off of the nitrous compounds, it interfered in a fashion which made it impossible for the test to act properly. The Abel heat test, which was the one they generally used, was the standard test for British cordite. The Abel heat test, which depended on the starch paper being affected by the emanations given off from the cordite under heat, was masked by the mercuric chloride, which prevented the emanations from being given off. Coming to the point which had been much discussed, could it be said that the mercuric chloride was useful in cordite, or served any other purpose than masking the heat test? He emphatically said no. He quite agreed that in the case of gun-cotton, on the contrary, mercuric chloride was used for a useful purpose. Gun-cotton, unlike gelatinised cordite, was not a hard, horny substance, but had an open texture. Like everything else easily accessible to the atmosphere, it was apt to be ravaged by bacilli, which formed a fungus that destroyed the gun-cotton. Mercuric chloride, which was a very fine antiseptic, destroyed the bacilli and prevented the fungoid growth. The nature of cordite, however, excluded these fungoid growths, and there was not any excuse for using mercuric chloride as an antiseptic in cordite. In the case of cordite there was one purpose only for which, in the opinion of his advisers, mercuric chloride could be used, and that was for masking the heat test."—*Mining Journal*, June 15, 1907, p. 810. (A. McA. J.)

The Reduction of Carbon Dioxide to Formaldehyde.—"In connection with the assimilation of carbon dioxide by plants, it has long been a debated point as to how this oxide of carbon is reduced to form the carbohydrates present in the plant. Recently the view that formaldehyde is the first product of the reduction of carbon dioxide has come into favour, although there is but little direct evidence to justify it. Moreover, the reduction of carbonic acid to formaldehyde is a change which has hitherto eluded all laboratory methods. This reaction has now been successfully effected by Dr. H. J. H. Fenton, F.R.S., who has obtained appreciable quantities of formaldehyde by treating an aqueous solution of carbon dioxide with metallic magnesium; the proportion of the aldehyde is considerably increased if weak bases, such as ammonia, aniline, or aluminium hydroxide, are present. Formic acid may also be reduced to formaldehyde by the action of magnesium. The fact that these reductions have been carried out by purely chemical means affords experimental justification for the view that a similar reduction of carbon dioxide occurs in the green parts of plants under the influence of sunlight. (Proceedings of the Chemical Society, March 21, 1907.)"—*Times Engineering Supplement*, April 24, 1907. (J. A. W.)

The Study of Crystals.—"It has long been known that if a saturated solution of a salt be protected from disturbance it may become very much supersaturated before it crystallises. A well known curve representing the solubility of a substance in a solvent is obtained by taking temperature for ordinates and concentration for abscissæ, any point on the curve showing the amount of the substance which the solvent can hold in solution at a particular temperature. Professor Miers finds that when a supersaturated solution—that is, one holding more than this amount—is stirred in an open vessel, a thin shower of crystals appears at the saturation temperature, but that nothing further happens until a temperature about 10° lower is reached, when a second and very much denser shower occurs. If, however, the experiment is made in a sealed tube, only the second shower falls, and no amount of shaking will induce crystallisation at the ordinary saturation temperature. The explanation is a remarkable one—namely, that the first shower only falls because the liquid becomes inoculated with crystal germs floating about in the air, while the second shower is due to spontaneous crystallisation.

Prof. Miers finds further that the second or 'labile' shower corresponds to a second curve more or less parallel to the ordinary solubility curve and below it, which he calls the supersolubility curve, and the interval between corresponds to a condition called 'metastable,' the terms being due to Prof. Ostwald, who predicted but never verified these phenomena. The somewhat disconcerting fact of the presence of such crystal germs in our truly wonderful atmosphere suggested to Prof. Miers the idea of experimenting with two substances, salol and betol, so rare that it could hardly be expected that their germs could be floating about. The experiments succeeded admirably at first, and both the curves of each were worked out by him and by his assistant. But towards the close, the very work itself had impregnated the air, and metastable crystallisation by inoculation began to occur. It also suggested to him some researches in association with Mr. Barker, which resulted in discovering that, although it was not necessary that the germ should be a minute crystal of the substance itself, yet it must be one which is either isomorphous with it, or is able to form parallel growths with it. The conditions requisite for parallel growth of one crystal on another have been established; the essential being that the structural dimensions of the crystals of the two substances shall be very nearly alike. The constants defining these internal stuctural dimensions are known as 'topic axes,' and were independently and simultaneously employed for the first time by Dr. Muthmann in Germany, and Dr. Tutton in this country, and they depend simply on the molecular volume and the axial ratios of the crystals. One of the most beautiful experiments shown was that of small crystals of sodium nitrate crystallising on calcite, two substances which have almost exactly the same topic axes, and which therefore fit into each other precisely."—Prof. H. A. Miers. —*Times Engineering Supplement*, May 8, 1907. (J. A. W.)

DETERMINATION OF FREE ACID IN SUPERPHOSPHATES.—"The following method of determining free phosphoric acid in superphosphates is both rapid and accurate. Five grams of the material, dried for three hours in the water-oven, are digested for three hours at the ordinary temperature with 100 c.c. of ether in a stoppered flask, the mixture being shaken at intervals. Fifty c.c. of the filtered extract are evaporated to dryness, the residue is taken up with water, and the solution, filtered if necessary, is then titrated with $N/4$ sodium hydroxide solution, using methyl orange as indicator. The yellow colour appears as soon as all the phosphoric acid has been converted into mono-sodium phosphate."—J. VAN DORMAEL, Bull. Soc. Chim. Belg., 1907, *21*, 103—115. *Journal of the Society of Chemical Industry*, April 30, 1907, p. 424. (A. W.)

STEAM IN GAS PRODUCER PRACTICE.—"This paper, presented by Dr. W. A. Bone and R. V. Wheeler, described experiments carried out with a gas producer plant at the iron and steel works of Messrs. Monks, Hall & Co., Warrington, with a view to determining the influence of variation in the proportions of air and steam in the blast upon the composition of the gas, its suitability for furnace operations, and the general and thermal efficiencies of the producers. Arrangements were made for the accurate determination of every possible factor in the results, and the authors believe that the results may be considered as establishing reliable data on the subject. The principal results of five trials are summarised in tabular form and go to show that the quality of the gas obtained steadily deteriorated as the steam saturation temperature was raised beyond 65° C., and that, whereas at 60° and 65°, the average percentage of total combustibles in the gas exceeded the high figure of 47 per cent., it steadily dropped to about 42 per cent. as the steam saturation temperature was raised to 80°. It is true that the falling off in quality was partly counterbalanced by an increase in the quantity produced per ton of coal gasified, but the general efficiency of the process fell off as more steam was used. With a steam saturation temperature of 60°, the resulting gas contained 27·3 per cent. of carbon monoxide, 16·6 per cent. of hydrogen, 3·35 per cent. of methane, and only 5·25 per cent. of carbon dioxide. At a saturation temperature of 80° the percentage of carbon monoxide had fallen to 16·05, the hydrogen had increased to 22·65 per cent., and the carbon dioxide to 13·25 per cent. In this connexion the paper points to the higher furnace efficiency of a producer gas rich in carbon monoxide in comparison with a gas of equal calorific value in which carbon monoxide is partly replaced by hydrogen. From the point of view of thermal efficiency the results tabulated show that the use of steam beyond that required to saturate the air blast at 60°, was not attended by any increased economy of working, and the thermal efficiency fell from 7·15 per cent. to 60·4 per cent. as the steam saturation temperature was raised from 60° to 80°. If a gas producer be regarded primarily as an apparatus for ammonia recovery, it should be worked with the highest steam saturation temperature consistent with the production of combustible gas, but if ammonia recovery be considered in conjunction with the suitability of the gas for furnace purposes, and the thermal efficiency of the process, the best steam saturation temperature is possibly between 65° and 70°. At 65°, over 30 per cent. of the nitrogen in the coal can be recovered as ammonia, which is nearly double the amount usually recovered in by-product coking processes, and, at the same time a gas is produced very suitable for furnace purposes.

Mr. Robert Armitage, M.P., said that the conditions under which the author's work was carried out were not normal working conditions, and that the production of a certain amount of saturated gas under the conditions described with an air blast saturated at 60° C. was not a guarantee of any degree of success in everyday work. To run regularly under such conditions would result in the shutting down of the plant, and the usual practice where steady and continuous working was desired was to use more steam than corresponded with saturation at 60°. The most surprising feature of the experiments was the attempt made to gauge the usefulness of the gas in the furnace without cooling it. He had been running a similar plant at his works for six or seven years, and while the Mond gas, rich in hydrogen, was found to be a very excellent heating agent, the necessity for cooling was fundamental.

Mr. B. H. Thwaite said the investigation had sealed the fate of hydrogen as an economic agent in power-gas production. The perfect power gas contained 30 to 32 per cent. of carbon monoxide, and not more than 7 per cent. of hydrogen. It would be interesting to know what potential power capacity of plant would justify the putting down of a by-product plant on the super-saturated or Mond system. Mr. A. H. Lymn disagreed that a power gas should not contain more than 7 per cent. of hydrogen. Engines were running on Mond gas which contained up to 28 per cent. of hydrogen without trouble from pre-ignition. Further than this, Messrs. Ehrhardt and Sehmer were putting down two 3,000 h.p. engines to run on coke oven gas containing 54 per cent. of hydrogen. The results shown in the paper had been arrived at by working the plant under abnormal conditions. It was not surprising that poor results were obtained with the larger quantities of steam. Mr. Walter Dixon said he had been engaged in investigating the same problem, and the practical result of the paper was that if you wanted a good gas very little ammonia should be expected, and if plenty of ammonia was required it was useless to trouble about the gas. Experiments which had been made on a commercial scale rather pointed in the same direction as the conclusions stated by the authors."—Iron and Steel Institute.—*Times Engineering Supplement*, May 22, 1907. (J. A. W.)

METALLURGY.

GRINDING IN TUBE MILLS.—"The ore contains a large proportion of hard, chalcedonic quartz, and the gold exists in an exceedingly fine state, conditions which necessitate very fine crushing in order to obtain a high extraction of the precious metal. The results of grinding in tube mills in other countries were so satisfactory that three tube mills were erected at the Waihi mill, the installation being completed in May, 1905, since which time an average duty of about 2·7 mills has been maintained. The mills are of the Davidsen, 22 ft. type, and are run at a speed of 27·5 revolutions per minute. Each mill is loaded with 5·5 tons of flints, and requires 50 h.p. to operate it. The quantity of flints consumed is 18 cwt. (2,016 lb.) per mill per week. Various liners have been used, including silex and Delarue quartzite blocks, and also cast iron liners, 1·25 in. thick. The iron liners last about as long as the quartzite blocks, namely, 2·5 months, but the grinding result is not so good. A new lining, invented and patented by H. P. Barry, called the 'honeycomb lining,' is now being tried with very promising results. This liner consists of a light cast iron frame, 22 by 14 by 3 in. deep, shaped to the curve of the mill. Thin walls divide this lining into four or six compartments. A temporary sheet iron back is fastened to the frame, and each compartment is then firmly packed with rough lumps of hard quartz or quartzite, varying in size up to 4 in. square bedded in with a mixture of Portland cement, coarse sand and fine sand. The liners so formed are allowed to set, preferably under exhaust steam, for several weeks—the longer the better—before being placed in the mill. This method of lining calls for a much shorter stoppage than with the quartzite blocks. The frames fit neatly with each other and with the shell of the mill, and only a small quantity of cementing material is required. If made with hard material these liners stand very well, and cost, including labour, about $175, as compared with $400 for lining with quartzite blocks. The grinding efficiency of a mill with this new liner appears to be quite equal to that of the quartzite blocks. Costs of running the tube mills are given by the writer.—Bi-Monthly Bull. Am. Inst. of M.E., January, 1907.—E. G. BANKS.—*The Mining World*, March, 1907, p. 306. (H. A. W.)

THE HOMESTAKE SLIME PLANT.—"The expenditure of half a million dollars by the Homestake Mining Co., of Lead, S. Dak., in the construction of a plant designed to treat material carrying gold values of an average of about 80 cents per ton, would seem to mark the opening of a new era in metallurgical economics.

The plant in question now nearing completion, is located at Deadwood, S. Dak., and will treat by cyanidation, by pressure lixiviation in filter presses, the slimes resulting from the daily crushing by stamps of some 4,000 tons of ore. The pulp from the stamps, after plate amalgamation, is separated by water classification into some 2,300 tons of leachable material or sands, which is reserved for the cyanide leaching plants, the remaining 1,700 tons, in the form of unleachable slimes, have heretofore been run to waste.

For some two years past Mr. C. W. Merrill, of the Homestake Mining Co., has been at work on the slimes problem, his solution of it taking form in a new design of filter press hereinafter described, which he has perfected and patented. The salient feature of his design, aside from great capacity, is the use of an automatic sluicing device by means of which the press can be quickly emptied of its contents without being opened. By this improvement Mr. Merrill proposes to reduce the labor cost of pressure filtering to practically nothing, whereas by the old method of opening and emptying the press by hand, this feature constitutes the principal item of expense.

The Homestake company is at present dropping 1,000 stamps, of which number 640 are at Lead, near the southern end of the company property, 100 are at Central, some 2 miles distant and near its northern boundary, while the remaining 260, at Terraville, occupy an intermediate position.

The location of the different mills with regard to the topography of the country, results in a natural division of them into two groups, termed northern and southern, respectively; each group being tributary to one of two adjacent water courses, each of which conveys a small creek. These gulches converge, and on steep grades eventually join at Deadwood, some 4 miles below Lead.

The mills at Lead, three in number, form the main or southern group and operate in conjunction with Cyanide Sands Plant No. 1, located in the gulch just below the town. The mill at Central, and the one at Terraville, together form the northern group and operate in conjunction with Cyanide Sands Plant No. 2, located at Gayville in the gulch some distance below the mills. In the selection of a site for the new plant to treat the residues from the two groups, the junction point of the two gulches offered ideal conditions.

Classification.—The classification of the pulp from the different mills, is as follows:—

The pulp from the mills at Lead has a consistency of about 12 parts water to one part solids, a sizing test on the solids giving the following results: Coarse (remaining on 100 mesh), 22 per cent.; middles (between 100 and 200 mesh), 18 per cent.; fines (passing 200 mesh), 60 per cent.

This pulp, after passing the plates, flows by gravity to what is known as the upper cone house where it is evenly distributed to 16 gravity settling cones; these are of iron, 10 ft. in diameter at top and with sides sloping at 50 deg. The overflow from these cones (overflow No. 1) consists of about 30 parts water to 1 part solids, all solids readily passing 200 mesh; the overflow, or spigot product, passes immediately, by gravity, to a second set of 12 cones, similar to the first, except that they are smaller, being 7 ft. in diameter. The overflow from this second set (overflow No. 2) has about the same character as overflow No. 1; the spigot product enters a 12 in. cast iron pipe line and on minimum grade of 2·5 per cent. is transported for a quarter mile down the gulch to Cyanide Sands Plant, No. 1. Here it is distributed to a third set of nine cones, identical with those of second set, overflow from which (overflow No. 3) further assists in elimination of water and slime material; the underflow, which is now quite thick, passes immediately to 36 cone classifiers, the product of each settling cone being distributed to four classifiers. These last are 3½ ft. in diameter and are equipped with a special design of feedwater inlet for discharging sands and admitting the water, an invention of Mr. Merrill. These classifiers throw off another portion of water and slime material (overflow No. 4), while the spigots furnish a product which, though carrying a considerable proportion of fines, is readily percolable and passes directly to the sand tanks.

The slimes which have, by different steps in the classification, been carried over in the overflows are in a very dilute condition, about 30 parts water to 1 part solids, all solids being under 200 mesh. Economy in its subsequent treatment, as well as conservation of water, requires the unwatering of this material to a large extent, this being accomplished by means of large clarifying tanks. Overflows No. 1 and 2 are joined and conveyed by pipe line to a group of 18 of these tanks, nine of which are 18 ft. in diameter and 20 ft. deep, the other nine being 26 ft. in diameter and 20 ft. deep. They are built of California redwood and have vertical sides, containing however, false cone-shaped bottoms whose function is to prevent any sudden slide of solid slimes into the spigot opening below. Being evenly distributed to these tanks, the thin slime pulp assumes a state of relative quiescence and to a large extent drops its burden of solid material which settles to the bottom and is drawn off with enough water to give it easy transport. The overflow is comparatively clear water and is discharged into a pond immediately below, formed by damming a portion of the gulch, from which it is subsequently drawn for sluicing purposes at the new plant at Deadwood, as well as for other purposes in connection with the company operations. The combined spigot product enters another pipe line and by gravity passes down the gulch to sands plant No. 1, being there joined by overflows No. 3 and No. 4, the combination passing to a second set of clarifying tanks, six in number and identical with the larger ones of the previous set. These throw off another large portion of water in nearly clear condition, which is used in the above sand plant. The underflow from these tanks contains practically all the solids of the combined overflows and has a consistency of about 3 parts water to 1 part solids, in which condition it is conducted by a 12 in. cast iron pipe line, on minimum grade of 1½ per cent., a distance of some 3 miles to the new plant at Deadwood.

A glance at the flow sheet will show that the scheme of treatment of the Terraville pulp differs but slightly from that of the southern division. The only real point of difference being the omission of the intermediate set of settling cones, corresponding to the difference in volume of pulp. There is also a slight difference in dimensions of some corresponding parts which in other respects are identical. The mill at Central sends its pulp directly to settling cones at Sands Plant No. 2, its course being easily traced on the flow sheet.

The final slime product emerging from this division corresponds closely with that coming from the other and is conveyed by gravity, in a 10 in. pipe line, for a distance of about two miles to the plant at Deadwood.

The two pipe lines, bearing material from the northern and southern mills, converge and eventually discharge into separate sludge storage tanks at the slimes plant.

The Filter Press Plant.—This new plant occupies four main benches, excavated from the side hill and faced, where necessary, by concrete retaining walls. The uppermost bench carries the two sludge storage tanks, before mentioned, identical with the larger clarifying tanks used in the classification, also a third tank of the same dimensions placed alongside for the storage of wash water. Directly back of and over the sludge tanks is the equipment for furnishing lime, in the form of emulsion, to the pulp as it passes to the presses below. Thirty feet below and some distance in front of this top bench is another carrying the solution storage tanks, two in number, for weak and strong solution, each being 32 ft. in diameter and 14 ft. deep. On a floor directly over these tanks are placed the two precipitation filter presses. Another 30 ft. below and in front is the main or press bench on which are placed the 24 large filter presses; while directly in front of and 24 ft. below these is the lowest bench, carrying the two precipitation or weak solution tanks, 22½ ft. diameter by 18 ft. deep; two standardising or strong solution tanks of the same size as the last; also a large waste-water storage tank; solution pumps; compressor; boiler, etc.; also the assay department in a separate concrete building. This bench directly overlooks the creek.

At some distance back from the main portion of the plant and at an elevation of 150 ft. above the press bench is placed the sluicing water storage tank having a capacity of 100,000 gallons. This receives its supply of water through a 10 in. tile pipe line connecting it with the overflow dam at Lead, previously mentioned, and in turn furnishes water, under sufficient head, to accomplish sluicing operations in connection with the presses below. From the 10 in. main which leads this water down the hill, a side connection leads off to the wash water storage tank on the upper bench, the supply being automatically regulated by a float.

The pulp is drawn from the centre of the bottom of each sludge tank into a 10 in. pipe. These pipes merge at once into a single 12 in. main which passes down the slope to a point above and midway of the line of presses, where it divides at right angles to form a straight 10 in. line running both ways as far as the two end presses. From this line, 6 in. laterals lead off at right angles, the line of each lateral paralleling and being equidistant from two adjacent presses, to each of which it delivers the pulp through flexible hose connections. The necessary amount of lime emulsion is added to the pulp just at the point of its exit from the sludge tanks.

*The Filter Presses.**—In most respects these immense presses resemble an ordinary press of the flush-plate and distance-frame type, the only salient point of difference in their operation being the use of the automatic sluicing device already mentioned. In fact it is upon this feature that the economic success of the entire scheme of slime treatment may be said to hinge.

In its larger details a press consists of an outer supporting frame, 91 flush plates, 92 distance frames, a follower head with thrust screws, together with the necessary connections for entrance and exit of solutions, pulp, etc.

The supporting frame consists of massive front and rear standards, spaced 46 ft. apart, between faces, and joined by side rails made from 15 in. channels. Between these rails are suspended the plates, frames, and follower head, each of these units being cast with projecting side ears which transmit the weight of the piece of the rails through roller bearings. Owing to the great weight of the assembled units and the necessity of preventing any sag or bulge, the side rails are provided with two intermediate, adjustable supports which, extending upward, are also tied across the top.

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* See this *Journal*, p. 174, vol. vi.

In general outline the plates and frames are similar, being rectangular in form, and measuring 4 ft. by 6 ft., outside. Each plate has a 2½-in. circular opening through each of its four corners, also two other circular openings on its vertical centre line, the upper one being 4 in. and the lower one 6 in. in diameter. With the exception of a narrow plane area surrounding these openings, and continuing as a border around the outer periphery, the entire surface of the plate is corrugated by channels.

The two opposite faces of a plate are identical, and communication between their corrugated portions is afforded by six small openings, thus insuring an equalisation of pressure on the two faces. Two small openings at the side corners also communicate directly with the adjacent circular channels through cored holes in the plate rim; these will be termed solution holes, as it is through them that the solutions pass to and from the slime cake.

There are three types of frame used in a press, known respectively as standard, top-feed-and-bottom-discharge, and top gate frames. They are all 4 in. in depth and have circular openings, symmetrical with, and of the same size as the corresponding ones in the plates, also the plane peripheral border, so that when any number of plates and frames are brought into close contact, these openings make continuous channels extending from end to end of the nest of units. In any frame, however, the centre bottom opening is not continuous through a full thickness of the frame rim, as the latter is slotted in the thickness of the metal from the top to the bottom of the opening to allow play to the sluicing nozzle, later described. The centre bottom opening simply passes through two semi circular guards, each ½ in. thick, leaving the slot 3 in. wide in the centre.

The top-feed-and-bottom-discharge frames, besides the openings already mentioned as common to all, each have two bottom corner discharge outlets, a centre, bottom, pulp, discharge outlet, also a centre, top, pulp inlet. In all frames, communication between the inner space of the frame and the top central opening is established by mean of three small cored holes.

The standard frames are similar to those just described, with the omission of the top feed and bottom discharge openings, while the top gate frames are of the standard type, with the addition of two gates in the top rim through which the interior space of the frame may be examined at any time. The front standard and follower head are also provided with top pulp inlet and bottom pulp discharge openings, their inner faces conforming exactly to those of a plate.

The number of each type of frame used in a press, together with their arrangement, is such as to provide, including those of the front standard and follower head, six pulp inlets, ranged along the top centre line, six pulp discharge openings below, also four discharge openings along each lower side.

In assembling the units of a press, the plates and frames alternate, each plate being so turned, relative to its neighbours, that the solution holes of any two adjacent plates alternate as regards the right and left side of the press. Thus, the even numbered plates will communicate through these holes with the circular channels on one side, and the odd numbered ones with these on the other side of the press. A strip of No. 8 canvas is hung over each plate, covering its two faces, circular openings in cloth coming opposite the corresponding ones in the plate. The assembled units, backed by the follower head, are then brought firmly together, the canvas strips serving as so many gaskets, and held in position against the front standard by two large thrust screws extending from seats in the follower head, back through brass nuts set in the rear standard, and terminating at the rear of the press in capstan heads, worked by hand levers.

Automatic Sluicing Device.—Extending throughout the upper portion of the lower central channel is a 3 in. pressure pipe, the rear end of which bears in a seat in the follower head, while the front end passes out through a stuffing box in the front standard, and directly connects with the water supply coming from the sluicing water tank on the hill above.

Projecting into each frame compartment from this pressure pipe is a small nozzle which may be directed towards any portion of the compartment by turning the pipe about its axis. The excess in size of the channel over that of the pipe leaves a crescent-shaped opening from end to end of the press, the same being continuous through the plates but opening directly upward into each frame compartment, and also having the six discharge openings below. All external communication to this space is closed at all times, save when sluicing out the slime cake.

Cycle of Operations.—The press is filled by admitting pulp from the overhead lateral main, through hose connections with the six feed openings, all of which will be opened by one movement of a lever, into the upper central channel, whence it passes downward into each frame compartment through the cored holes; the pulp entering the press under a gravity pressure of about 35 lb. per sq. in. During the filling of any compartment, the water of the pulp finds its way through the canvas bounding faces, along the plate corrugations back of them and out into the corner channels through the solution holes. Thus, all four channels may be utilised for discharging this water, but ordinarily the two top ones will be closed, and all effluent water will pass into the lower ones, and find exit through the eight discharge openings into 5 in. collecting pipes, paralleling each lower side of the press, from which it will be finally discharged. This water, which is perfectly clear, will ordinarily be run to waste, but in case of shortage it may be led to the large waste water storage tank on the lowest bench, and from thence pumped back to the sluicing water storage tank on the hill above. The solid portion of the pulp is retained in the press and ultimately fills all compartments, the pressure of filling rendering any artificial method of packing the cake unnecessary.

The press being filled, the cakes are subjected to six hours' treatment, consisting of lixiviation, oxidation, and washing, in exactly the same manner as open vats are treated, except that all operations must be conducted at higher pressures than in vat percolation.

The period of six hours for all treatment in the press has been determined to be the best for Homestake slimes, but would vary with the nature of the slimes in the case of other ores, probably being less in some cases and more in others. Bearing in mind the arrangement of the plates, it is evident that the corner channels on either side of the press may be used for entrance of solution, etc., those on the opposite side serving for the discharge of same.

In order, however, that any liquid may pass through the press via these channels, any portion of it must force its way through a cake and its cloth bound faces. Usually but one channel on a side will be used, the other being closed.

Sluicing Out the Slime Cake. Preliminary to this operation the six discharge openings along the lower central channel are opened, all other channels being closed. The discharge openings will be controlled by the operator through one motion of a lever. Water from the sluicing water tank is then turned into the pressure pipe under a pressure of about 50 lb. per sq. in., and issues from each nozzle as a powerful jet direct against the cake in each compartment. By means of a small capstan, fixed to the pressure pipe at the head of the press, it is first turned backward and forward a few times by hand through a gradually increasing angle, in order to loosen the cake directly around the nozzle and clear the bottom channel. This preliminary breaking accomplished, the motion throughout an arc of some 200 deg. is continued automatically by means of a pinion set on the pipe and gearing with a rack, which derives a reciprocating up and down motion from a pitman, whose upper end has a projecting roller engaging a groove cam developed on the face of a cylinder, which is revolved about its vertical axis by gearing from an electric motor. This cam device travels bodily on an overhead track, one machine serving eight presses.

The slime as it is broken down from each compartment is washed between the semi-circular guards, into the central crescent shaped channel surrounding the pressure pipe, whence it passes along and finds exit from the press through the nearest bottom discharge opening, the progress of the operation being gauged by the density of the escaping pulp.

The amount of water necessary for sluicing operations is about 4 tons per ton of slimes; but with the use of two nozzles for each frame, it will be reduced to 3 and possibly 2½ tons, of which all but 1 ton or less may be pumped back and re-used after first settling in tanks or dams.

Each press has a capacity of about 25 tons of dry slimes, and the time required for a complete cycle of operations is about 8 hours divided as follows: filling 1 hour, treatment 6 hours, and sluicing out 1 hour. Thus each press has a capacity of about 75 tons of slimes per 24 hours, or a total for the plant of between 1,700 and 1,800 tons per day. One man has complete charge of at least four presses.

Recovery of Gold and Circulation of Solutions.—The solution drawn from the strong solution storage tank and used in the first lixiviation operation, leaves the presses as weak solution, and having been discharged into the precipitation or weak solution tanks below, is pumped up to the third bench and forced through the precipitation filter presses. As this solution enters the suction pipe of the pump it receives a carefully gauged supply of zinc dust, in the form of emulsion, brought down from the surface through a small pipe. The zinc becoming thoroughly mixed with the solution, precipitates most of the gold before the presses are reached. These presses, also of a design patented by Mr. Merrill, have plates and frames of a triangular form. The solution with its burden of precipitates enters the press via a channel in the upper corner and is led downward to the apex of each frame through a small channel cored in the side rim. By thus introducing the solution into the frame from below and taking into consideration the shape of the compartments, Mr. Merrill proposes to get a constant and more intimate contact of the entering solution with any undissolved zinc. The precipitates being retained in the frame compartments are permitted to accumulate for some time, or until the frames are filled when they are removed by hand, cupelled, melted down and refined.

The clear solution escapes through the cloth-bound faces of each compartment and leaves the press through a row of cocks near the lower line of the press, each cock communicating with the corrugated faces of a plate. This solution now passes to the weak solution storage tank beneath the floor, from which it again passes to the presses below as weak solution. Emerging from a press in a still more weakened condition, it is now run to the standardising or strong solution tanks below, where it is brought up to strength by fresh additions of potassium cyanide and from here it is pumped up and discharged into the strong solution storage tank, thus completing the circuit.

Two electrically driven triplex pumps, size 8 in. by 6 in., serve to elevate the solutions.

Compressed air is furnished by an electrically driven duplex compressor, size 22 in. by 18 in.; and steam for heating all the buildings is supplied from a 250 h.p. vertical, water tube boiler.

The power necessary to run the plant will be about $\frac{1}{10}$ h.p. per ton of slimes.

Several of the presses are completed and have been in commission long enough to demonstrate their efficiency in all respects, an extraction of 90 per cent. being authoritatively reported, thus verifying the results and predictions deduced from the 10 ton trial press which was operated over a year and treated some 6,000 tons of slimes.

The total cost of treatment will be in the neighbourhood of 25 cents per ton.—MARK EHLE, Jr., E.M. —*Mines and Minerals*, March, 1907, p. 358. (J. Y.)

EXTRACTION OF GOLD FROM REFRACTORY ORES. —"A patent of J. W. Worsey and E. Hoal (846,768, March 12, 1907) describes a process primarily intended for extraction of gold from refractory ores, but stated to have a wider application. The ore is first crushed and screened and freed from aluminium silicates in any known manner. It is then 'subjected to the action of nascent bromide of chlorine produced in the body and presence of the ore by mixing with the powder or crushed ore chlorate of soda and bromide of soda, or a chlorate and bromide of other equivalent alkali or a chlorate and bromide of an alkaline earth.' The mixture is quietly stirred and some acid is added, and the temperature is raised from time to time until it rises to about 150 deg. F. After this treatment has been carried out four hours, the temperature is raised to near boiling point for the purpose of completing the dissolving of the gold and finally to free it from excess of gaseous products. The whole is then allowed to settle; the clear solution is decanted away and the residue is washed with hot water. The solution and the wash water are mixed together, and any acid present is neutralised by means of a quantity of a solution of an alkali. A weak solution of a lead salt is then added and the whole vigorously stirred, and H_2S is added to the solution with proper stirring (by blowing air through) for 10 to 20 minutes. The whole then turns quite black, and in this condition the precipitated gold and lead sulphide settle out, and are collected, dried, and the gold recovered by a smelting operation. To recover any residual gold contained in the ore previously separated from the gold solution, it is treated and washed in a vat with a weak solution of double or single ferro-cyanide of ammonium and commercial soda carbonate* after the manner of ordinary leaching. The effect of this is that all the residuary gold in the ore is dissolved out of it, and this is effected quickly, namely, within 24 hours. The gold is dissolved out in the first stage as bromo-chloride, and afterwards when heated it resolves itself into auric chloride, $AuCl_3$. In this treatment the ferro-cyanide of ammonium acts as an expeditor or saver of time, and by it the residual ores of this and other gold extraction processes can be economically treated and gold extracted, and within a greatly reduced time. The solution resulting from this leaching operation can be treated like the previous solution with sulphide of hydrogen (H_2S) for the separation of the auriferous lead sulphide."—*Electro-chemical and Metallurgical Industry*, April, 1907, p. 149. (J. A. W.)

MINING.

Preservation of Mine Timber from Decay.—"The Philadelphia and Reading Coal and Iron Company, and the United States Forest Service, in February, 1906, planned a series of experiments to be conducted along practical and economical lines, with the primary idea of saving timber and money. Actual tests have been carried on to discover just what methods of handling and treatment would give the greatest service at the least expense. The factors destructive to timber used by anthracite coal companies are (1) decay, (2) breakage, (3) wear, (4) insects, (5) waste. Sets of round gangway timber, used in the main haulage ways, were tested as a basis for this experimental work.

In the experimental work the following were the main points studied:—(1) The benefits of peeling timber; (2) the benefit of seasoning timber or drying it out; (3) the benefit of treating timber with various preservatives applied by several processes. The preservative treatments tested and their relative costs are given in the accompanying table:—

COST OF PRESERVATIVE TREATMENT OF TIMBER.

Method of application.	Preservative.	Cost per cub. ft. Dols.
Preservative heated to 180 degs. Fahr. and applied in two coats with a brush.	Creosote (dead oil of coal tar) ...	0·01½
	Avenarius carbolineum ...	0·04½
Immersion in an open tank without pressure—successive baths of hot and cold fluid.	Solution of common salt 15 per cent.	0·02
	Solution of zinc chloride, 6 per cent.	0·03½
	Creosote (dead oil of coal tar) ...	0·11
In a closed cylinder under vacuum and pressure.	Solution of zinc chloride, 6 per cent.	0·07
	Creosote (dead oil of coal tar) ...	0·15

Ventilation is a large factor in the destruction of timber by decay. In badly-ventilated air-courses, where there is sufficient moisture and temperature, decay is at a maximum. For wood-destroying organisms to exist, they must have air and water. If it were possible to keep the timber always wet or always dry, it would not decay; it is the alternation of wet and dry conditions which produces rot.

By direct experiment, it has been definitely proved that in durability, peeled timber is superior to unpeeled timber. In unpeeled timber the space between the bark and the wood favours the development of wood destroying organisms and furnishes a breeding place for many forms of insect life. Peeled timber is 7 to 8 per cent. lighter than unpeeled timber. Great benefits may be derived from seasoning the timber or drying it out. Seasoned timber is far more durable than green timber, and is considerably stronger. A period of from two to five months is necessary for proper seasoning. In that length of time it may lose from 30 to 35 per cent. of its green weight. In case the company handles its own timber from the woods to the mines, the possible saving in freight due to peeling and seasoning may easily be estimated.

Of the treated timbers, those preserved with creosote and a solution of zinc chloride by the open tank method have been very successful. A good penetration of the wood by the preserving fluid from 2 to 5 in. has been easily secured, and timbers so treated effectively resist decay. Timbers treated by the cylinder process are also standing well, but in order to be a profitable investment they will have to show a considerable increase in life over those treated by the far less expensive open-tank process. The success of the superficial brush treatment has yet to be proved. Timber so treated is not as effectively resisting decay as the other treatments. On account of their comparative cheapness and ease of application, requiring no plant, they may be of benefit to small operators or in a situation where the timber is likely to be broken. Acting on the results obtained from this series of experiments, the Philadelphia and Reading Coal and Iron Company is considering the advisability of taking up treating work on a larger scale. Plans have been drawn for the construction and erection of an open tank treating plant at one of its collieries. If erected, either creosote or zinc chloride solution will be the preserving fluid used.

The successful application of a preservative treatment to mine timber is largely dependent upon the regulation of the timber supply of the consumer, and a timber supply that will admit of this regulation. Under these conditions, timber cannot be rushed from the woods into the mines. There must be time for treating it and for preparing it for treatment. This means the storage of a reserve supply of felled timber at one or more points.

It should be distinctly understood that this series of experiments especially applies to the southern anthracite coal region, where the Pennsylvania and southern pines are the leading mine timbers. To regions where conditions and species of timber differ, the results of these experiments may not be applicable."—J. M. Nelson (Lecture).—*Colliery Guardian*, May 24, 1907, p. 961. (A. R.)

Observations on the Future of New Zealand Dredging.—"Perusal of the New Zealand Mining Handbook, 1906, furnishes information on various points affecting the future of New Zealand dredging. Collected together, the observations gleaned from the various reports indicate the need of more powerful dredges, of systematic prospecting, and of improvements in gold-saving appliances, more especially for fine gold.

In these fields, in the opinion of Mr. R. McIntosh, Inspector of Mines, and author of the prize essay on the Mineral Resources of New Zealand, greater constructional strength, capacity, and power are needed to deal with bank claims, and to further the work of gorge dredges.

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* Cyanide (?)—Ed. Comm.

On the one hand, the bank claims are now the principal scene of operations: and for these, large, powerful dredges are required, as an enormous amount of low grade or worthless material has to be shifted to secure the auriferous wash. On the other hand, since 1900, the gorges of the various rivers have afforded the most numerous instances of dredging failures. In the gorges many obstacles are met, such as hard bottom, tight wash, travelling drift, and short seasons, all of which call for increased power.

Recent discoveries have emphasised this need, for the breaking through of a hard crust or false bottom at Waitri, on the Karawau River, and at Waenga, on the Clutha River, with the disclosure of rich underlying wash, suggests that many dredges have hitherto been working on this false bottom, leaving the under wash, to attack which more powerful machines are necessary.

The enforced employment of a larger and heavier type of dredge will have, as offset to its higher initial cost, the advantage of increased durability. By the adoption of machinery suitable for deep, heavy ground, the life of dredges in the most successful locality, the large basin between Clyde and Alexandra, has been indefinitely prolonged. The same remark applies to the powerful dredges successfully working in the Clutha Basin, around Lowburn. Large capacity dredges, moreover, are needed where, as at Big Waikaka, Southland, the ground must be turned over rapidly to secure good returns.

The importance of systematic prospecting in the development of the dredging industry is frequently accentuated by the District Wardens. The extensive flats and swamps of Southland, says Mr. McIntosh, offer a large field for exploitation. It is difficult to say where suitable gravels do not exist. Systematic boring, in the hands of capable operators, is the cheapest and easiest method of testing these gravels. Systematic drilling, such as has been carried on at Waikaka with reliable results, would prove the existence or otherwise of an extensive dredging field on the Upper Nevis. And again, the opinion is still held by those who know the Cardrona field, that the whole length of the valley—a distance of over 20 miles— will pay for working, provided the extent of auriferous wash is located by boring rods, and suitable dredges used. There is an enormous area of dredgable ground in the Mataura Valley, but only by systematic boring can the payable areas be located.

The problem of gold, more especially of fine gold, recovery is an ever-recurring one, and the solutions still leave room for improvement. From the many reports, writes Mr. McIntosh, as to the payable prospects obtained from the (Matakanui) claims, before the dredge commenced to work, I should be inclined to think that our present method of treating the wash obtained in dry land dredging is not sufficiently advanced. And elsewhere, as a dredging field, the (Lake George) portion of the (Waiau) district has great possibilities, provided suitable appliances are available for saving the fine gold"—C. C. LONGRIDGE. —*London Mining Journal*, April 13, 1907, p. 486. (J. Y.)

MISCELLANEOUS.

THE NEW ENGLISH PATENTS BILL.—"At a meeting of the Society of Chemical Industry held at Burlington House, Mr. R. J. Friswell, who presided at the meeting, read a paper on the Patents Bill of 1907. He pointed out the possible hardship of a rigid interpretation of the demand for samples in a chemical invention. The draftsmen appeared to think that chemical samples were equivalent to mechanical drawings, whereas they were quite different. If samples were to be deposited it was necessary to settle their fate. We knew very little of the stability of chemicals. Many classes of aldehyds, many cyanogen compounds, many dyestuffs, and even some hydrocarbons, nitro compounds, azo compounds, nitrites, and sulphites were notoriously unstable, and being accustomed to use chemicals freshly prepared, very little was known as to the stability of the remainder. What was the object of keeping the samples? Were they to be used against the patentee in the event of litigation? It was generally undesirable to separate chemical patents into a separate class from patents in general. Clause 2 distinctly did this. The clause as to compulsory licences was likely to lead to increased litigation, and thus to be as costly as heretofore. The words 'adequate extent' are retained in the Bill. How is working in the United Kingdom to an adequate extent to be determined? If these actions became frequent a new Court would be required. At any rate, the Comptroller was not a good authority. Working of foreign-owned patents should be compulsory, or the Act of 1883 should be left alone. Clause 9, opened the way to very vexations proceedings against a patentee. He might be working his patent when a trade rival might apply for revocation, thus forcing him to compromise or to face litigation. This clause was as much at the disposal of powerful syndicates as of any one else. At any rate, existing patents should be relieved from the operation.

A letter from Mr. Levinstein was read. He objected to a paper by Mr. Bloxam and Mr. Imray, and urged the necessity of controlling the foreign owners of patents in the interests of British industry. Mr. Bloxam urged the undesirability of compulsory working. It had become a farce abroad. If we took this attitude the Colonies would follow and patentees might be expected to have to work in all Colonies or forfeit their patents. Mr. Salomon urged the need to allow easier means of obtaining extensions of meritorious patents. Often much of the life of a valuable patent goes in getting it to work; a trivial patent often succeeds at once and had a long life. The returns on a patent should not be the sole criterion as to its extension. He was impressed by the chairman's words as to the instability of samples, but this could be met by requiring their analysis. Mr. Guttman opposed compulsory working. After remarks from Mr. Hugh Moulton and others, Mr. Astbury, M.P., said he had been greatly interested by the paper. The chairman alone appeared to have seen the true bearing of many of the clauses. Mr. Lloyd-George was animated by a single-minded desire to help British industry. It was not too late to improve the Bill and he was sure that the suggestion as to analysis of samples would be of use, and Mr. Pearce's suggestion of the insertion of words making it certain that the samples were only for the use of the Comptroller was valuable. He had also been struck by the same point as the chairman had urged—the undesirability of the putting of decisions as to adequate working on the Comptroller. He would not be able to keep counsel, experts, and so on in order, as well as a Judge. He hoped that that would be altered before the Bill took its final form.

The chairman summed up. He said that all were agreed on the undesirability of the deposit of samples.

Some differed on the compulsory working clauses, and he thought that as to the rest there was practical unanimity. He was glad to see that Mr. Astbury was at one with them as to the difficulties of the proposal as to the Comptroller becoming a kind of Judge, and to hear that the revocation clause had been modified."—*Times Engineering Supplement*, May 15, 1907. (J. A. W.)

NEW ENGLISH PATENTS BILL.—"The second reading debate on the Patents and Designs Bill in the House of Commons leads one to apprehend that the compulsory working clause will be passed substantially in the form in which it has been drafted. For reasons, some of which are given below, I fear that the effect of this clause would on the whole be detrimental to the interests of this country, and I therefore hope that it will be given further and very careful consideration before the passing of the Bill.

Many other countries have somewhat similar provisions under which patents may be revoked unless they are 'worked,' *i.e.*, their inventions carried out, within a given time in the countries in question.

The experience gained from the working of such laws may be gathered to some extent from the opinion expressed by the International Association for the Protection of Industrial Property, which holds congresses from time to time in different countries and consists of a large number of experts, including patent barristers, patent agents, and manufacturers, of various nationalities. This association, at its congress in Berlin in 1904, passed, by a large majority, a resolution affirming that the omission to work a patented invention shall not lead to the revocation of the patent, but to the granting of compulsory licences. This resolution was confirmed at the congress at Milan in September, 1906.

As a result of the compulsory working law in France, where probably the law in respect to the working of patents is as stringent as anywhere, important manufactories have, indeed, been established, but not only have they been established and worked by foreigners, to whom their profits have been sent abroad, but, although a native industry was previously flourishing, the foreign competition thus created has obtained the command of the market. The manufacture in question is a chemical one, and it is well known that it is in regard to such manufactures that the demand for compulsory working has arisen in this country.

In Canada, I am informed, the difficulty of getting inventions taken up and manufactured within the time allowed is found to be so great that many large companies who are so near at hand as in the United States, no longer take out Canadian patents.

It may be said that it is rather to our advantage that the patents of foreigners, against which the present Bill is directed, should be declared void. Is not this, however, a short-sighted view? The late William Siemens once said that it was rarely that any success attended inventions that were not the subject of monopolies; and he added, "If I found an invention in the gutter, I would rather give it to one man than to the public, for in the latter case it would be practically lost."

One effect of the proposed alteration of the law seems probable—that is, as in the case of Canada, foreign inventors will cease to take out British patents, and particular in the case to which the proposal is specially directed—namely, inventions relating to dye-stuffs—and that foreign manufacturers will simply keep their processes secret. Consequently the technical knowledge of this country will not be enriched, as at present, by the published specifications of the patents in question.

Another result will probably be that not only the Colonies but the United States will, in some way or other, act in retaliatory manner—perhaps by simply following in our own steps.

Now, the disadvantages above indicated do not attach to the plan of compulsory licences, the proper carrying out of which would be the true solution of the problem; for not only would it enable inventions to be worked in this country whenever such working were really required, but it would still preserve the patents for the inventions and so, according to the quoted opinion of Siemens and the general experience, do much to ensure the successful establishment of the invention in the industry of this country.

It is true that compulsory licensing has been tried in the country and failed. Its failure, however, has been due to the expense of the procedure. Now, the present Bill provides an inexpensive procedure for obtaining revocation—namely, by placing the power to revoke in the hands of the Comptroller, subject to a single appeal—and I submit that if, instead of this, the Comptroller were given the power to grant a licence to any suitable applicant to work a patent that, for a given time after its grant, had not been worked in this country, a satisfactory solution would be found to a really difficult problem."—*Times Engineering Supplement*, May 15, 1907. (J. A. W.)

PRINCIPLES UNDERLYING THE AMALGAMATION OF GOLD IN THE STAMP MILL.—"Amalgamation is a physical—not a chemical—process, the catching of gold by mercury being almost instantaneous. While this is true with respect to a given particle of gold, yet, with respect to the whole mass of the ore, it is a continuous process.

The purpose of wet-stamping or crushing is to free the gold from its adherent gangue, the pulp, as it passes through the battery-screens, consisting of particles of gangue and particles of free gold, which, mingled with, say, eight times their weight of water, flow over the mercury-coated apron-plates in a film, or layer. The metal alone is capable of being wetted by the mercury, the ore particles of low specific gravity not being at all affected. The gold, as soon as it comes in contact with the amalgamated surface, is immediately wetted by it, just as ordinary substance is wetted by water. The action is assisted by the fact that gold and silver, owing to their greater specific gravity, soon come in contact with, and are wetted and caught by, the mercury of the amalgamated surface. If the particle of gold is less in diameter than the thickness of the mercury film, it produces no disturbing effect upon it; if greater, it forms a projecting point.

A liquid, such as this mercury film, has a tendency, due to surface-tension, to provide itself with a horizontal upper bounding surface. In the attempt to do this, in that part of the surface which has been raised up because of the gold grain lying beneath, the pull of the surface-tension, resolved into its components, tends to press down the grain upon the plate with considerable force. This force begins the instant the grain of gold is wetted by the mercury, continuing to operate as the grain is drawn beneath the surface, and constituting the so-called 'attraction' of the mercury for the gold. Because of this force, grains of gold adhere to each other and to the plate with great tendency, especially where the pulp drops upon it.

Gold telluride, or gold alloyed with bismuth, becomes incapable of being wetted by mercury, and so is not caught. Again, in ores which have been subject to alteration, the grain of gold may be coated with oxides, sulphides, or base metals, so as to be incapable of being wetted. In one case the gold grains were coated with a film of chalcedonic silica.

Soluble salts in the ore react with mercury, causing it to be lost in solution and precipitating other elements in it, tending to make it 'sick.' This sickness is of two kinds; another metal may become alloyed with the mercury, making it less active, or more viscous and brittle, so that flouring may readily be induced. The fine slime also adheres more readily to the surfaces of sick mercury, and the metals in it frequently re-oxidise, forming a coating which prohibits the coalescence required.

The finer the gold the more rapidly does the mercury take it up, so that more mercury is needed to amalgamate fine gold than coarse.

The chief difficulty in the use of plain copper plates is due to the fact that a small amount of copper from the plate goes into solution in the mercury, and this is easily attacked by oxidising agents, and by soluble salts of the ore, forming a tarnished film of copper-salts on the surface of the mercury, and preventing the gold from being caught The difficulty is overcome by silver-plating the copper. Silver is less soluble in mercury than copper, and is less affected by the soluble salts abovementioned, and not at all by oxygen. The effect of silver-plating on the absorption of mercury by the copper plate is to restrain it at first, since the mercury has to diffuse the silver; eventually, however, the amount absorbed is approximately the same.

From a physical standpoint the temperature of amalgamation does not matter as long as it remains constant. A disturbance of plate-equilibrium arising from changing temperature is detrimental, since, in such a case, the thickness of the mercury film, as well as the absorption of mercury by any other metal, changes; and the millman then has to deal with varying conditions. Where chemical action is negligible, it might be advisable to conduct amalgamation at higher (but constant) temperatures, since the wetting of the gold particles by mercury is slightly facilitated with a rise of temperature; but a comparatively low one is better where the influence of soluble salts in the ore has to be considered."—(*Mining and Scientific Press*) *New Zealand Building, Engineering and Mining Journal*, Jan. 28, 1907, p. 118. (W. D.)

TO KEEP IRON FROM RUSTING.—"There are many causes of the rusting of iron. It may be produced by atmospheric action alone, but in a majority of cases galvanism plays a large part in the destruction of the metal. Long experience has shown how rapidly iron nails employed in fastening sheets of lead and copper upon roofs are destroyed, the other, the electro-negative metal, remaining comparatively unaffected. The electrolyte, or exciting fluid, which by acting on the iron and not on the other metal, or by acting more upon the former than upon the latter, causes the electric current, is either actual water from rain or snow or the water vapor always present in the atmosphere. The decomposition of the water causes the liberation of oxygen at the positive pole, which is the iron, and this nascent oxygen rapidly combines with the iron. Now, it is claimed that red lead is an excellent material for protecting iron from rust and electrical action. Unfortunately, however, red lead is more electro-negative to iron than either copper or lead. Hence, should moisture by any chance get between the red lead and the iron, the destruction by rust is more rapid than when iron is in contact with copper or lead. This electro chemical action is at the same time strengthened by the purely chemical action between the red lead and the carbonic acid always present in the air, an action which converts the red lead into ceruse, whereby an additional quantity of nascent oxygen is set free to rust the iron. It is also highly probable that the carbonic acid has an independent action upon the iron, thereby much facilitating its oxidation. It must not be forgotten that every porous place and still more every crack in the paint becomes sooner or later an entrance for water and carbonic acid. A good oil varnish is by far the best protection for iron, but it must, of course, be properly used. Not only must the iron be scrupulously, practically, and chemically clean and dry when the varnish is applied, but the covering must be without a flaw. Varnish will not adhere to greasy, rusty, or wet iron, and the contraction of the varnish on drying will cause minute cracks at such places and the iron-destroying gases will find their way through these cracks and get between the iron and the non-adherent varnish. Again, the varnish must be thoroughly dry before the iron is exposed to the weather. The varnish may be coloured if the colour does not interfere with the strength, continuity, and elasticity of the protecting skin, but is best dispensed with."—*(Architectural Art) Mining and Scientific Press*, April 13, 1907. (W. D.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 265/07. J. McConnochie. Improvements in apparatus for concentrating diamondiferous or metalliferous material. 26.6.07.

(P.) 266/07. G. A. Sheeley. Improvements in prospecting drills. 26.6.07.

(P.) 267/07. A. M. Day. Improvements in means for distributing pulp in cyanide tanks and the like. 27.6.07.

(P.) 268/07. A. C. Whitehead. Improvements in machines for washing and concentrating alluvial and other diamondiferous material. 27.6.07.

(P.) 269/07. A. D Wilson. An improved method of separating under water pressure precious minerals, metals and other substances from lighter materials or soils. 27.6.07.

(P.) 270/07. W. Charles (1), A. Lumley (2). Improvements in means for preventing overwinding. 27.6.07.

(P.) 271/07. C. H. N. Williams. Improvements in the treatment of copper ores. 27.6.07.

(P.) 272/07. A. Harvey. An improved gold and diamond saving machine. 27.6.07.

(P.) 273/07. J S. Walker (1), T. A. Walker (2), E. R. Walker (3). Improvements in appliances for steadying, controlling or buffering valves of air engines or other mechanism liable to sudden deleterious shocks. 28.6.07.

(C.) 274/07. J. Ashworth. Improvements in lift pumps. 28.6.07.

(P.) 275/07. H. Bell (1), J. Bell (2), H. Airth (3). Engine rock drill. 28.6.07.

(C.) 276/07. E. S. G. Rees. Improvements in rotary condensers and in condensing steam turbines. 28.6.07.

(C.) 277/07. H. S. Rankin. Improvements in valves. 28.6.07.

(C.) 278/07. H. Herrenschmidt. Improvements in or relating to the extraction of metals and metalloids from their ores or compounds. 28.6.07.

(C.) 279/07. C. H. Haeseler (1), A. H. Taylor (2). Improvements in portable pneumatic motors. 29.6.07.

(C.) 280/07. C. H. Haeseler. Pneumatic tool. 29.6.07.

(P.) 281/07. J. van Waart. A device or construction to create perpetual motion by force of gravitation or by magnetism or by both combined. 29.6.07.

(P.) 282/07. D. Mills (1), A. J. Irvine (2). Improvements in or appertaining to steam boiler furnaces. 1.7.07.

(P.) 283/07. B. G. Richter. Hand cultivator. 5.7.07.

(C.) 284/07. A. O. Tate. Apparatus for treating liquids electrolytically. 5.7.07.

(P.) 285/07. A. J. Arbuckle (1), A. Osborne (2). Improvements relating to the generation of steam for working rock drilling machines, engines, pumps and the like in a mine and apparatus therefor. 5.7.07.

(C.) 286/07. C. A. Parsons. Improvements in and relating to blading for turbines, compressors and the like. 5.7.07.

(P.) 287/07. J. Temple. A new or improved counting frame or like calculating device. 6.7.07.

(P.) 288/07. W. E. Bleloch. Improvements in means for recovering diamonds or other precious stones or gold or other metals from loose matter. 6.7.07.

(P.) 289/07. T. R. Lewis. A gauge for holding the wires of a fence in position while inserting plain wire picket which has to be twisted up. 8.7.07.

(P.) 290/07. K. E. Fawns (1), G. Kermode (2), R. J. Dickie (3), J. H. Brown (4). Improvements in coin-freed machines for vending postage stamps, tickets and the like. 8.7.07.

(P.) 291/07. R. B. Ballantine. An improved process and apparatus for rock boring or drilling in mines and the like. 9.7.07.

(C.) 292/07. J. Fuchs. An automatic conveyance for returning the balls used in playing the game of billiards or similar games that may be played on a billiard table having pockets for the reception of such balls as are driven off the table in play by the striker to the strikers or baulk end of the table. 10.7.07.

(C.) 293/07. R Pickard. An improved taximeter for cabs. 12.7.07.

(C.) 294/07. H. J. Rabone. Improvements in flexible steel or metal measuring tapes. 12.7.07.

(C.) 295/07. A. Hayes. Improvements in or relating to the manufacture or treatment of iron or steel. 12.7.07.

(C.) 296/07. J. B. Moyet (1), H. Bouvier (2). Improvements in automatic couplings for railway wagons and the like. 12.7.07.

(C.) 297/07. A. H. Gibson. Improvements in and relating to percussive machines operated by reciprocating fluid columns such as electro-pneumatic track channellers. 12.7.07.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

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Milroy, A. T., *l/o* Liverpool; *c/o* Lyman Sons and Co., St. Paul Street, Montreal, Canada.

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Robertson, R. H., *l/o* Germiston; New Jersey Zinc Mines, Franklin Furnace, New Jersey, U.S.A.

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THE JOURNAL

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VOL. VIII. AUGUST, 1907. No. 2.

Proceedings
AT
Ordinary General Meeting, August 17, 1907.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, August 17th, Prof. J. Yates (President), in the chair. There were also present :—

38 Members: Messrs. T. L. Carter, F. Alexander, R. G. Bevington, W. R. Dowling, K. L. Graham, A. Heymann, A. McA. Johnston, A. Richardson, Prof. G. H. Stanley, H. A. White, Prof. J. A. Wilkinson, W. Beaver, G. Bernfeld, E. H. Croghan, N. M. Galbraith, G. Goodwin, J. Gray, F. G. Guthrie, W. H. Jollyman, J. A. Jones, J. Kennedy, C. B. Kingston, A. Kressmann, J. Lea, C. W. Lee, D. McKerrell, G. Melvill, J. T. Milligan, P. T. Morrisby, D. J. Pepler, O. D. Ross, W. Sharp, H. Taylor, J. A. Taylor, W. Taylor, J. P. Ward and F. W. Watson.

8 Associates and Student: Messrs. J. H. Harris, R. W. Leng, R. Lindsay, C. A. Robinson, C. Toombs, W. Waters, L. J. Wilmoth and J. Innes.

12 Visitors and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as published in the *Journal*, were confirmed.

NEW MEMBERS.

Messrs. Gray and Croghan were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been duly elected, as follows :—

BALLANTINE, ROLLO BOWMAN, 11 and 12, Steytler's Buildings, P. O. Box 1136, Johannesburg. Civil Engineer.

EVANS, A. W., French Rand G. M. Co., Ltd., P. O Box 25, Luipaardsvlei. Assistant Surveyor.

METCALF, JOSEPH ERNEST, P. O. Box 209, Germiston. Chemist.

WIGHT, A., Randfontein Estates and G. M. Co., Ltd., P. O. Box 2, Randfontein.

The Secretary announced that His Excellency the High Commissioner had been pleased again to accept the Hon. Presidency of the Society, and that the Council had also elected the following new Hon. Vice-Presidents: Mr. de Villiers (Minister of Mines), Mr. Reyersbach (President of the Chamber of Mines) and Mr. George Albu.

The following gentlemen had been admitted Associates by the Council since the last general meeting.

BROWN, WILLIAM WAUGH, Mysore G. M. Co., Ltd., Marikuppam, Mysore State, South India. Chief Reduction Officer.

GERARDY, BARON GEORGE, Geldenhuis Deep, Ltd., Cleveland.

MAXWELL, ROBERT KERR, *c/o* Mr. J. Hiddleston, P. O. Box 140, Fordsburg. Chemist.

NEWLAND, NOEL NELSON, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanider.

RHODES, CLARENCE EDGAR, El Oro Mining and R.R. Co., Ltd., El Oro, Mexico. Cyanide Manager.

THOMAS, WILLIAM, Champion Reef G. M. Co., Ltd., Mysore State, South India. Assistant Cyanide Chemist.

Mr. McArthur Johnston: I beg to propose a very hearty vote of thanks to Mr. Rowland for the excellent index which accompanied the last issue of the *Journal*.

The President: I am sure we are all in sympathy with the motion. It is only when one comes to arrange an index oneself that we realise the amount of work it entails.

PATENTS PROCLAMATION AMENDMENT BILL.

The President: The proposed Patents Proclamation Amendment Bill has already been brought to your notice. You remember that the Government some time back appointed a Commission to investigate the system of administration of the Patents Office and other matters pertaining to patents and designs. The Commission took evidence and its Report has just been issued. May I place a few of their conclusions before you:

(*a*) In the interest alike of inventors and of the public it is undesirable to restrict the grounds on which oppositions may now be entered to an application for a patent.

(*b*) As stated in the Public Service Commission, the principle of having a trained officer to deal with such oppositions in the first instance is a sound one.

(*c*) There is not sufficient work arising in the Transvaal Patent Office as at present constituted to keep an officer fully occupied on his present work only.

(*d*) It is highly desirable that there should be uniformity of Patent Law throughout South Africa, and if that were achieved circumstances would warrant the appointment of a Commissioner of high scientific and legal attainments to carry out for South Africa the functions at present performed by the Commissioner in the Transvaal.

(*e*) The abolition of the office of the Commissioner of Patents with his present duties and qualifications would be a retrograde step from the point of view of the public and inventors alike: if, therefore, the above proposal (*d*) is absolutely impracticable, it is desirable that the Commissioner of Patents in the Transvaal should also be available as legal adviser to the Attorney General.

(*f*) If neither of the two previous proposals (*d*) and (*e*) are possible, patent oppositions in the Transvaal should be heard in the first instance by a Judge of the Supreme Court, subject to an appeal to the full Court.

(*g*) In that event the officer in charge of the Patent Office should be confined to administrative duties. He should, however, still be a gentleman with legal training, and the office should remain, as at present, a separate department.

You will gather from the above that the Commission, with the exception of Mr. Juta, advocates, and I am justified in saying, strongly advocates, the retention of a Commissioner of high scientific and legal attainments, but in the face of this finding, and in opposition to the unanimous opinion of Scientific Societies in Johannesburg, the Government has introduced, and yesterday finally passed, a Bill providing for the withdrawal of the judicial functions of the Commissioner of Patents, thus lowering this official's status, and transferring the judicial functions to a Judge of the "expensive" Supreme Court.

The Attorney-General assured the Legislative Council on Thursday that the Government had acted largely on the advice of Mr. Bucknill, the late Commissioner of Patents, but why they should ignore the finding of the Commission specially appointed to investigate the subject is difficult to understand; there would appear to be a direct saving, by the new arrangement, of about £1,000 per annum in the department, but as a debit against this we have, in the opinion of many competent to judge, such additional trouble and expense thrown upon inventors as will deter many from coming forward, and this will lead to a loss of revenue, both directly and indirectly, altogether out of proportion to the direct saving mentioned. And this is quite apart from the question of the advisability of having a really highly trained technical man to deal with opposition in the first instance, a point emphasised by the Commission.

There is, however, some satisfaction to be derived from the Attorney-General's statement to the effect that the Government propose to deal more fully with the whole subject of designs, trade marks and patents next year, and that it hopes to come to some arrangement with the other South African Colonies regarding these matters, and realises that, in the event of such an arrangement being arrived at, it might be advisable to have one man in charge; we are also given to understand that the present legislation is merely temporary.

We lodged an objection to the proposed Bill some weeks back, but without avail, and this week, in conjunction with the South African Association of Engineers and the Institute of Mechanical Engineers, we wired a request that the Government should receive a deputation from the Societies concerned, before finally passing the Bill. No heed was paid to our request, but we are indebted to several of the Honourable Members of the Legislative Council for taking the matter up and for assisting us in our endeavours to prevent, what we believe to be, retrogressive and injudicious legislation.

We take very strong exception to the opinion of one of the Hon. Members of the Council that the views of the Scientific Societies of this Colony should not be considered in such matters. Our member and the members of the sister Societies are pre-eminently the men who have to deal with inventions and patents, and their views are therefore entitled to respect, a respect which has hitherto not been denied them. If the Government will not accept the advice of its own Commission, and that of men whose business it is to deal with patent matters, whose advice will it accept, and by whom will it be guided?

The Bill being finally passed we can only hope that next year will bring forth legislation of a more satisfactory nature.

Mr. Fred. Rowland: I am requested to express the regret for the absence to-night of representatives of the S. A. Association of Engineers, who, owing to various unforeseen circumstances are unable to be present. I am, however, authorised to state that they are in entire agreement with the action we have taken in this matter in protesting so emphatically at the action of the Government in ignoring the recommendations of their own Commission and in hurrying through Parliament a Bill which will tend to increase the cost, despite the statements of the Attorney-General and Mr. Juta to the contrary, of patenting inventions, should there be any opposition, in this Colony.

In order to show you how hurriedly this matter has been dealt with, I beg to submit the following facts. You will remember that at our June meeting, after certain information came to our knowledge, a resolution expressing the views of the Society was passed. The resolution was forwarded to the Attorney-General and also to a number of members of the Legislature on the 19th of June, and on the 27th June a reply was received from the Attorney-General acknowledging its receipt and referring us to the reply given to Dr. Macaulay on the 25th June, which stated that the matter was receiving the careful consideration of the Government. We heard nothing further from the Government on the subject, and as we had no idea of their intentions, it was not possible for us to make any further representations. On the 2nd August the Attorney-General stated in the Assembly that on the 6th inst. he would ask for leave to introduce the amending Bill. The Government Notice, No. 866, accompanying the Bill emanated from the Prime Minister's office, and is dated the 2nd inst.

The Bill was published in an *Extraordinary Supplement to the Government Gazette* on the 3rd inst. (Saturday). Monday was a Public Holiday and even had we known of its existence it was practically impossible for us to have obtained a copy before the 6th inst. I might here mention, that it was only after the application by Mr. Chaplin, on the 11th July, that the Report of the Patents Office Administration Committee was laid on the table of the House on the 16th July. Even at this date the authorised Government issue of the Report will not be available for the public, so I am informed, for about another fortnight, although extracts have been published in the local press. The second reading of the Bill was carried after a protest had been made by Mr. Chaplin, who, I understand, acted in consultation with Dr. Macaulay in the matter, on the 12th inst., and the third reading on the 13th inst. The Bill went to the Upper House on the 14th inst. and was finally passed, as you have seen, on the 15th inst., altogether fairly rapid progress. The Attorney-General complains of our action in telegraphing to members of the Council and states that we should have written or consulted members on the subject. I might mention here that our wire was not the cause (as the Hon. Mr. van der Merwe stated) of the business of the Upper House being wasted for a whole day. As the result of our resolution I may inform you that several Hon. Members had gone very thoroughly into the matter, and although it was not reported, the subject was very thoroughly debated in the Council, and the points we had raised brought well before the notice of the Government. Your thanks are due to the Hons. W. Dalrymple, W. A. Martin, T. A. R. Purchas and R. Feetham, for the very thorough manner in which the matter was dealt with, as also to the Hon. Mr. Robertson for his reply to Mr. van der Merwe. I need hardly say that it is a very difficult matter to obtain a representative meeting of the Councils of the several Societies at short notice, members of which are busy men and situated all along the Reef, but there was ample time between the end of June and the beginning of August for the Government to have asked and received the opinions of all those interested in this Bill, as used to be done in the past. There has really been no time in this instance for our having an opportunity of going into the Bill and making further representations to the Government.

The Attorney-General has been advised that the compilation of an Index of Transvaal Patents would cost £10,000. He also states that the number of patents applied for annually is very small. If the latter statement is correct, the cost of this index seems enormous. The Hon. Mr. Wolmarans rightly mentioned that we objected to the Bill on the grounds of the increased cost, but he is further reported to have said, we should pay the increased cost, a statement which will not appear encouraging to the inventor, who, as a rule, is only able at present to pay the existing charges with an effort. After our existence of 13 long years, during which period the press of this city have given much publicity to our work, we find an Hon. Member unaware of this Society's existence. I have taken the opportunity of enlightening him on this subject, and I am sure you will endorse the cordial invitation, which I have sent to him to attend our meetings.

Mr. A. C. Whittome (Past President of the Institute of Mechanical Engineers): It gives me great pleasure on behalf of the Transvaal Institute of Mechanical Engineers to support the vigorous protest you are making against the action of the Government. I think the Hon. Member of the Legislative Council who was unaware of the existence of this Society ought to be placed in our museum as one of the curiosities of the Transvaal. He must be the only public man in the country who is unaware of it. Together with your Association, the Transvaal

Institute of Mechanical Engineers, to which I have the honour to belong, has done its best to prevent this Bill being hurriedly passed through all its stages in both Houses of Parliament. As you know, we made a great effort at the time the Commission on Patent Office Administration was sitting. We laid our views before the Commission and pointed out that so far from taking any retrograde step, the Government should progress, that they should give us more than we had had in the past, and all that we are entitled to from our representative character. The Hon. the Attorney-General stated in his remarks the other day that there was not a single individual, who could say that he had been treated with scant courtesy or consideration. I almost think that in saying this the Hon. gentleman was playing with words. Surely the Councils of the three big Associations here, representing over 1,700 technical men of the Rand, surely, they have not only been treated with scant consideration but practically with no consideration at all. It may be perfectly correct that individual men have received every consideration, but certainly the interested parties on the inventor's side have received practically nothing at the hands of the Hon. the Attorney-General or the Government. It appears that the inventor is always to be the man who receives last and least consideration. Those of us who have had anything to do with opposed applications for patents know that even since the war the cost of opposition is tremendous. Under the new regulation the opposition is to be heard before the Supreme Court, and the cost will be very considerably increased. Some gentlemen have gone so far as to say that the cost will be ten times the amount it used to be: this is probably too high an estimate, but it will certainly be 100 per cent. higher. Yet these inventors are the men who are helping to build up the industries of the Colonies. It was pointed out in the evidence before the Commission that a saving of 10d. per ton by improved methods in the gold mining industry alone meant about £62,000 to the Government per annum in profits tax, yet for a paltry £1,000 a year they have handicapped the men who have saved, many times over, that gain. I hope this Association will, and I am sure my own Association will, continue their efforts to get this ridiculous Bill withdrawn from the statutes of the country. We are told apparently as a last thought of the Hon. the Attorney-General, that it is only a temporary measure. Possibly they have something in reserve for next year which will be even worse than this, but I hope that the whole of the Scientific Associations of the Transvaal will continue their efforts and see that this time next year, if possible, we get not only what we had before but something better.

The President: There the matter stands, and the only satisfactory feature about it is the possibility of things being put right next year.

During the week I have received a letter from one of our Past-Presidents, Mr. Crosse, who is at present at Frankfurt, Pilgrim's Rest, and who suggests that part of it may interest you, so I will read it.

ASSISTING THE SOLUTION OF GOLD IN THE CYANIDE PROCESS BY COMPRESSED AIR.

By Andrew F. Crosse (Member).

"For many years I have been impressed with the idea of using compressed air, produced by a "trompe," for assisting in the solution of gold in the cyanide process. Time forbids my entering into a description of the "trompe," but I believe that a "trompe" was first used in the Pyrenees, during the middle ages, for iron smelting. In this district (Pilgrim's Rest) we have abundance of water-power, and I erected a small "trompe" on my arrival here last October.

More than half the gold here is contained in the slime, so that a high extraction from the latter is absolutely necessary. You are probably acquainted with the Pohlé pump, which is the exact opposite of the "trompe," the latter being an arrangement by which compressed air is formed, by being sucked down by water and delivered under pressure, whilst the Pohlé pump is an arrangement by which water, or any other liquid, can be lifted, under given conditions, owing to the reduction of its specific gravity, by the admixture of air. I make use of a conical vat, with a vertical pipe in the centre of the same, through which the pulp in the vat is lifted by compressed air, supplied by the "trompe," 2 ft. above the surface of the liquid; it falls over into an outer tube and is discharged through four pipes, having their orifices tangentially arranged so as to force the pulp into a circular motion. By this means I obtain full aeration and very rapid circulation. The solution of the gold is extremely rapid, and there is no machinery of any kind, it being absolutely automatic.

As this method was so very successful, I applied the same principle, slightly modified, to the sand treatment. The sand must be perfectly clean and free from slime, such as is obtained from the Wilfley concentrator. I have arranged a vat from which the solution drains off into an air-lift pump; this solution is circulated through the sand. After a few hours the sand is full of air bubbles, and a rapid circulation of the liquid is maintained. These air bubbles are formed owing to the cyanide solution being surcharged with air, and they are deposited on the sand, in the same way that a piece of bread, thrown into a glass of champagne, is coated with carbonic acid bubbles. I have obtained 92 per cent. extraction of the gold by this means from 6·2 dwt. tailings in 24 hours, including washing. This result is, I believe, a record."

The President said they were all pleased to hear from Mr. Crosse, and expressed the hope that he would soon contribute a paper embodying his experience in the Pilgrim's Rest district.

NOTES ON THE ESTIMATION OF CAUSTIC LIME.

By Edw. H. Croghan (Member).

I must, to a certain extent, ask for your forbearance for introducing this subject. You well know that the changes which occur on burning ordinary limestone are such, that besides water and carbon dioxide being driven off, ferrous carbonate and carbonate of manganese become ferric oxide and manganese dioxide, and organic matter is burnt or at least decomposed, leaving a small residue of carbon ; also, most important of all, the intermixed silicates are attacked, becoming so basic by combination with lime that their bases become soluble in hydrochloric acid, and their silica, on treatment with hydrochloric acid, is partly dissolved, and partly separates as metasilicic acid. Intermingled quartz undergoes scarcely any change in lime burning. On the other hand, if this quicklime be exposed to the action of the air, it reverts, gradually absorbing first water and then carbonic acid, and this reversion will be rapid or slow, according to the state of the atmosphere. Partly in consequence of this, it has always been and may possibly still be a vexed question, as to what is the best method extant for estimating the uncombined lime, if I may so call it, when it occurs admixed with its compounds and other constituents. In the title of this paper, it will be noted that the phrase caustic lime is employed. I adopt this expression in preference to such terms as "available lime" or "soluble lime," for in my opinion it seems a closer definition of this uncombined or free lime.

By "available lime" I should understand not only caustic lime, but also that existing as carbonate and other compounds, which would become available, as we understand the term here, if brought under suitable conditions. On the other hand, "soluble lime" is hardly a technical designation, being really used in a popular sense, and being therefore lax. Other expressions might possibly be suggested by some of our members and discussed.

Regarding the existing methods for determining caustic lime, I am not aware of any special literature which has tended to prove or disprove their accuracy. At present there seem to be two methods in vogue locally. One is the gravimetric and the second is the volumetric sugar process. Previous to the appearance of Mr. G. W. Williams' paper on "Notes on Lime, Clean-up, etc." before this Society, in which he recommended the sugar method, the gravimetric process was principally employed. At that period many chemists, myself included, were very sceptical as to the quantitative value of the sugar procedure. From my experience I found that the gravimetric process, *as usually calculated*, always yielded results much higher than those given by the sugar method. Therefore, with a view of trying to account for these discrepancies, I carried out a series of analyses and tests on certain burnt limes as supplied to the mines. Time did not permit me to go into all the detail I would have desired to, so I take this opportunity of submitting to you the various data I have obtained so far.

On receipt of the samples, they were quickly crushed and passed through a 60 mesh sieve. An average portion of each was taken and transferred to a small glass jar, the cover of which could be screwed down tight on to a caoutchouc ring, thereby ensuring complete protection from the air. Samples of burnt limes may be kept in this way for months without showing any appreciable deterioration. In all 9 burnt limes were completely analysed, namely, 8 white limes, Nos. 1 to 8 inclusive, and one so called "blue lime" No. 9.

The table referring to the analyses will be found on the next page.

For comparison these samples were also analysed by the sugar method, with the following results as to true caustic lime :—

1	2	3	4	5	6	7	8	9
%	%	%	%	%	%	%	%	%
63·00	75·50	71·05	63·35	74·72	73·85	64·40	70·35	30·80

The sugar process was carried out as follows : Two grammes of the sample were transferred to a litre flask, and then filled up to the mark with a sugar solution containing 20 gm. per litre.

Table I.

	1 White Lime.	2 White Lime.	3 White Lime.	4 White Lime.	5 White Lime	6 White Lime	7 White Lime.	8 White Lime.	9 Blue Lime.
Quartz and Insoluble Matter	3·48	2·23	2·22	5·80	2·63	2·61	2·82	3·32	6·11
Soluble Silica	0·51	0·66	0·40	1 05	0·75	0·73	0·51	1·18	1·54
Fe_2O_3	0·32	0 08	0·12	0·16	0·40	0·44	0·48	0·72	2 21
Al_2O_3	0·27	0·09	0·13	0·24	0.62	0·36	0·19	0·12	2·56
CaO (total)	77·32	85·84	83·88	75·32	84·00	83·76	74·36	82·20	44·10
MgO	0·91	1·02	0·93	0·68	0·78	0·95	0·59	0·85	21·71
MnO_2	nil.	nil.	nil.	nil.	nil.	nil.	nil.	nil.	2·79
SO_3	0·61	0·44	0·39	0·94	0·47	0·49	0 35	0·59	0·79
P_2O_5	trace.	trace.	trace.	trace.	trace.	trace.	trace.	trace.	trace.
CO_2	8·05	5·00	6·78	5·30	5·00	5·05	5·25	5·75	3·60
Moisture, Water of Hydration, Carbon, etc.	8·50	4 92	5·27	10 62	5·50	5 72	14·69	4·75	10·42
Total	99·97	100·28	100·12	100·11	100·15	100·11	99 24	99·48	98·86

The mixture was thoroughly shaken off-and-on for about an hour, allowed to stand overnight, shaken again for a minute or two, and an aliquot portion (250 c.c.) then taken and titrated by N.HCl employing phenolphthalein as indicator. The tests were carried out at least in duplicate and checked with themselves exceedingly well, the variation never exceeding 0·5 per cent.

Discussion of former methods of calculating caustic lime.—In the older gravimetric method of ascertaining the quantity of caustic lime, such detail as is given in the preceding analyses was not gone into. All that was essential was to estimate the total lime (calculated as CaO) and the carbon dioxide, and to calculate the latter into carbonate of lime, and subtract this from the total lime, the balance being assumed to be caustic lime.

On carrying out the calculation as above stated, the following would be the results for caustic lime:—

Table II.

	1 per cent.	2 per cent.	3 per cent.	4 per cent.	5 per cent.	6 per cent.	7 per cent.	8 per cent.	9 per cent.
$CaCO_3$ ..	18·30	11·36	15·41	12·04	11·36	11·48	11·93	13·07	8·18
CaO ...	67·01	79·48	75·25	68·58	77·64	77·33	67·70	74·88	39·52

Percentage error of old gravimetric method:—

Table III.

1 per cent.	2 per cent.	3 per cent.	4 per cent.	5 per cent.	6 per cent.	7 per cent.	8 per cent.	9 per cent.
6·46	5·27	5·91	8·25	3·90	4·71	5·12	6·44	28·31

Now, if these results are compared with those obtained by the sugar method, you will note that the caustic lime figures of the former are much higher than those of the latter. If the difference between the two methods on the figures here obtained be expressed in percentages of the caustic lime present as in above table (III.) it will be clearly seen that the error is a very serious one, especially in blue lime.

It, therefore, seemed to me that either (1) the sugar method is quite unreliable although checking with itself, or (2) in the case of the gravimetric process there are other factors that should be considered. Of course, it was also possible that both methods are incorrect. I was then somewhat nonplussed for some third method to decide between these suppositions, and eventually adopted the principle of one from Sutton's "Volumetric Analysis." As distinct from the "sugar" I shall term this the "acid" method. The process as devised and eventually carried out was as follows:—2 gm. of a sample is transferred to a vessel containing some 150 c.c. to 200 c.c. boiling distilled water, previously kept boiling some five minutes, and now boiled for a further two or three minutes, constantly stirring.

It is then kept gently simmering and titrated by *N*.HCl with phenolphthalein as indicator until disappearance of colour remains constant through half-a-minute.

Results of acid method:—

TABLE IV.

1 per cent.	2 per cent.	3 per cent.	4 per cent.	5 per cent.	6 per cent.	7 per cent.	8 per cent.	9 per cent.
63·42	75·74	71·47	63·63	75·04	74·76	64·68	70·70	Not Applicable.

On comparison of the figures with those of the sugar method, one sees them to be only slightly higher. This may possibly be due to the small amounts of oxide of magnesium present partly going into solution, and therefore being expressed as caustic lime. Sutton states the method is applicable in the presence of carbonates of lime and oxide of magnesium. My experience confirms the former, but certainly not the latter statement. In the case of the blue lime I could not apply the method, as not only was an excessive quantity indicated, (being far more than the total lime found), but the end point, owing to the suspension of the dark sample, was very difficult to see. As an experiment I added magnesia to one of the white limes, and found a much higher quantity of lime than before! Fortunately these eight white limes contained very small percentages of magnesia and, therefore, this method was quite applicable, the percentage differences from the sugar method being still plus, but quite small, thus:—

Percentage error of acid method.

1 Per cent.	2 Per cent.	3 Per cent.	4 Per cent.	5 Per cent.	6 Per cent.	7 Per cent.	8 Per cent.
0·66	0·32	0·59	0·44	0·43	1·23	0·43	0·50

It is evident therefore that in the absence of excess of magnesia, the two methods agree very closely. I came thus to the conclusion that the gravimetric method must be incomplete and therefore made the complete analyses given in Table I.

From the results there obtained it seemed evident to me that besides the CO_2 present, the consideration of the other acid radicals, such as SO_3, Al_2O_3, and soluble to (combined) silica might probably afford some clue to the higher figures calculated for caustic lime from the ordinary gravimetric process. I therefore carried out a series of calculations taking into consideration certain possible compounds of lime only. These compounds were as follows:—

(1) $CaCO_3$. (2) $CaSO_4$. (3) $Ca_3Al_2O_6$. (4) $Ca_2Si_3O_8$ (trisilicate) (partly fusible at temperature of assay furnace). (5) $CaSiO_3$ (bisilicate) (fusible at temperature of assay furnace). (6) $Ca_4Si_3O_{10}$ (sesquisilicate) (fusible at temperature of assay furnace). (7) Ca_2SiO_4 (monosilicate) (infusible). (8) Ca_4SiO_6 (basic silicate) (infusible).

Other aluminates or silicates may be formed, but the above are well known ones, and I therefore take them for the purposes of illustration and subsequent calculation of the caustic lime. I assume all through that magnesia, being a weaker base than lime, remains uncombined as MgO.

The various tables, giving the result of subtracting each lime salt in turn are to be seen on the next page: the figures are in per cent.

If the various data which have been obtained for caustic lime be now summarised, we shall be better able to compare them. They follow in the order of (1) gravimetric process and the subsequent deductions from the calculations, (2) the acid and (3) the sugar method.

With the last supposition, therefore, viz. (4 E), I have arrived at a theory which approximates really closely to the figures of the sugar and acid methods, being in fact sometimes higher and sometimes lower. If the differences between the two are stated in percentages, it will accordingly be seen how low or high the sugar method is:—

	1 %	2 %	3 %	4 %	5 %	6 %	7 %	8 %	9 %
High	2·23	1·41	3·13	0·41	—	—	1·29	—	—
Low.	—	—	—	—	1·64	0·20	—	0·68	5·81

The only high difference is that of the blue lime, No. 9, which is most probably due to the large amount of magnesia it contains, and there-

TABLE V.

	1	2	3	4	5	6	7	8	9
2. CO_2 and SO_3 into $CaCO_3$ and $CaSO_4$.									
$CaSO_4$ subtracted ...	1·03	0·74	0·66	1·60	0·80	0·83	0·59	1·00	1·34
CaO left	66·65	79·18	74·98	67·92	77·31	76·99	67·44	74·47	38·97
3. CO_2, SO_3 and Al_2O_3 (calculated as $Ca_3Al_2O_6$).									
$Ca_3Al_2O_6$ subtracted ...	0·71	0·24	0·34	0·63	1·64	0 95	0·50	0·32	6·77
CaO left	66·21	79·03	74 77	67·53	76·29	76·40	67·13	74·27	34·76
4A. CO_2, SO_3, Al_2O_3, as in (3), and soluble silica as $Ca_2Si_3O_8$.									
$Ca_2Si_3O_8$ subtracted ...	0·82	1·07	0·64	1·70	1·21	1·18	0·82	1·91	2·49
CaO left	65 90	78·62	74·53	66·88	75·83	75·95	66·82	73·54	33·81
4B. CO_2, SO_3, Al_2O_3, as in (3), and soluble silica $CaSiO_3$.									
$CaSiO_3$ subtracted ...	0·98	1·27	0·77	2·03	1·45	1·41	0·98	2·28	2·97
CaO left	65·74	78·42	74·40	66·55	75·59	75·72	66·66	73·17	33·33
4C. CO_2, SO_3, Al_2O_3, as in (3), and soluble silica as $Ca_4Si_3O_{10}$.									
$Ca_4Si_3O_{10}$ subtracted ...	1·14	1·48	0·89	2·35	1·68	1·63	1·14	2·64	3·45
CaO left	65·58	78·21	74·28	66·23	75·36	75·50	66·50	72·81	32·85
4D. CO_2, SO_3, Al_2O_3, as in (3), and soluble silica as Ca_2SiO_4.									
Ca_2SiO_4 subtracted ...	1·46	1·89	1·14	3·01	2·15	2·09	1·46	3·38	4·41
CaO left	65·26	77·80	74·03	65·57	74·89	75·04	66·18	72·07	31·89
4E. CO_2, SO_3, Al_2O_3, as in (3), and soluble silica as Ca_4SiO_6.									
Ca_4SiO_6 subtracted ..	2·41	3·12	1·89	4·97	3·55	3·45	2·41	5·58	7·29
CaO left	64·31	76·57	73·28	63·61	73·49	73·68	65·23	69·87	29·01

TABLE VI.

	1 Per cent.	2 Per cent.	3 Per cent.	4 Per cent.	5 Per cent.	6 Per cent.	7 Per cent.	8 Per cent.	9 Per cent.
(1) $CaCO_3$	67·07	79·48	75·25	68·58	77·64	77·33	67·70	74·88	39·52
(2) $CaCO_3$, $CaSO_4$ ...	66·65	79·18	74·98	67·92	77·31	76·99	67·44	74·47	38·97
(3) $CaCO_3$, $CaSO_4$, $3CaO.Al_2O_3$	66·21	79·03	74·77	67·53	76·29	76·40	67·13	74·27	34·76
(4) do. as (3) with $2CaO.3SiO_2$	65·90	78·62	74·53	66·88	75·83	75·95	66·82	73·54	33·81
(5) do. as (3) with $CaO.SiO_2$	65·74	78·42	74·40	66·55	75·59	75·72	66·66	73·17	33·33
(6) do. as (3) with $4CaO.3SiO_2$	65·58	78·21	74·28	66·23	75·36	75·50	66·50	72·81	32·85
(7) do. as (3) with $2CaO.SiO_2$	65·26	77·80	74·03	65·57	74·89	75·04	66·18	72·07	31·89
(8) do. as (3) with $4CaO.SiO_2$	64·31	76·57	73·28	63·61	73·49	73·68	65·23	69·87	29·01
(9) Acid	63·42	75·74	71·47	63·63	75·04	74·76	64·68	70·70	not applicable
(10) Sugar	63·00	75·50	71·05	63·35	74·72	73·85	64·40	70·35	30·80

fore the slight solubility of magnesia in lime sucrate solutions would naturally tend to slightly increase the apparent percentage of caustic lime. It is now evident from the agreement between the sugar method and the acid method, that the gravimetric process even with all these calculations cannot be considered to be particularly correct, the results from the above being erratic through the accumulation of experimental errors; whereas in comparing the acid and the sugar methods, the latter is too low in a very slight degree only, which can possibly be accounted for by the acid method dissolving small amounts of magnesium oxide.

I am therefore of the opinion that the sugar process is not only an excellent and quick technical method, but is also one yielding results possibly as accurate as can be got in our present state of knowledge even in the presence of much magnesium oxide, which is in reality not an error, because slightly higher results should be more than borne out in practice, owing to the slight protective action of magnesium oxide on cyanide.

In addition to the above I should also like to state the results of a few experiments carried out on one of the samples (No. 4) to see the effect of time of contact with the solvent. One gramme was transferred to a flask and made up to 1 litre with distilled water, shaken according to periods stated below, an aliquot part taken and titrated by *N*.HCl with phenolphthalein as indicator.

After shaking 15 minutes 60·2 % CaO was found
,, ,, 30 ,, 61·6 % ,, ,,
,, ,, 60 ,, 63·0 % ,, ,,
,, ,, 10 ,, allowed to stand overnight, and shaken again for a minute or so,
62·3 % CaO was found.

I do not lay any special stress on these experiments, but I simply wish to state the data obtained. Others may possibly be desirous of further experimenting. There seems no definite reason why, if practically CO_2 – free water be used, the best results should not be obtained by an aqueous extraction.

Before concluding, I wish to draw your attention to the so called "blue limes" and their general composition. My analytical experience has shown me that, as a usual thing, they are highly dolomitic limes, and sometimes even approach nearly to burnt dolomites. In addition they contain in practically all cases very appreciable quantities of manganese oxides.

A certain theory has, therefore, suggested itself to me about their utility. Those of you who have analysed the white limes as supplied to the mines, will have noted that their magnesia contents are quite small, not exceeding 2 per cent., and that they rarely contain manganese; if any, it will not be above traces or 0·1 per cent. Now it is most probable that these white limes deteriorate or revert at a very rapid rate to $CaCO_3$, whereas a so called "blue lime" would possibly revert very much slower, say, at a very small ratio to the amount of caustic lime present. Now under the present conditions under which these white limes are received, stored, and subsequently used, this may be of some practical importance. I am inclined to think that their large magnesia contents act as a partial protection and delay this reversion, the result being that, although a white lime may be up to the standard when it was tested, yet when ready to be used, it will not be of a much better quality in CaO content than that of a blue lime stored for a longer period. It, therefore, seems to me that under present conditions where white limes are exposed to the air, having merely a galvanized iron shed to protect them (which in my opinion is of little avail, since the whole affair has really a very free access to the atmosphere), that a blue lime which is much cheaper will do the same amount of work required as a white lime. The magnesia must not be forgotten, for although it is so insoluble—1 in 55,000 parts of water at 20° C. as compared with lime 1 in 800 parts of water at 20° C., *i.e.*, about 7 times as insoluble—yet it must play some part in consequence of the large volumes of liquid used. In addition we have also the manganese oxide contents, and these are possibly present in sufficient amount to cause an increase of extraction, without affecting the working conditions. It will therefore be of very great interest to hear the practical experience of those who may have used blue limes at any specific time. Also, perhaps some of our members might carry out some experiments, comparing the deterioration of white limes and blue limes under similar conditions, simply doing the assays by the sugar method.

There is just one other point, and that is, for those who try to purify their boiler-feed waters, never to employ these blue limes, but only the best white limes with as little magnesia present as possible. I mention this as I have been told of such blue limes being used, and thus making the purified waters in a worse condition than they were originally.

This paper is only a summary of various results that I have obtained, but I hope it will help in trying to solve this problem of determining caustic lime, and that not only through vigorous discussion but through others taking up the trend and experimenting, we shall be able to go further ahead.

The President: I have very much pleasure in moving a vote of thanks to Mr. Croghan for his paper. It has raised many interesting points and I hope we shall have a free discussion upon it. Lime has certainly become cheaper during the last ten years, but, nevertheless, during the twelve months ending June last year, the Transvaal mines spent about £64,000 on lime alone, so that it is not without an incidence on our working costs, and its use and testing are of interest.

Mr. McArthur Johnston: In rising to second the vote of thanks proposed by our President, I feel that we ought to congratulate Mr. Croghan on again raising this question, as the subject is a very important one to our industry. In buying lime we ought to rate it on the basis of caustic lime. About two years ago Mr. G. W. Williams brought this matter before us and practically proved that the sugar method was not only useful and quick but reliable. I was surprised to learn that Mr. Croghan only adopted the process some short time ago, as consideration easily shows that the gravimetric method is bound to give higher results than are available by solution of the caustic lime, which is, of course, the factor on which we buy.

That he tackled the question, however, in a very thorough manner his paper to-night proves. If I criticise Mr. Croghan's paper at all, I may say that he has failed to prove the presence in the lime of silicates and aluminates of lime. It is also to be welcomed in as much as it will ventilate a matter concerning which the lime burners are naturally very sore. To state the position as clearly as possible, we, that is, the consumers, desire to buy the lime soluble in water, and the higher that is, the better we are pleased, for two reasons, firstly reliability of quality saves the cyanide manager some little trouble, owing to the alkalinity being a constant factor: and secondly, the other ingredients are so much waste material to be handled.

Mr. Croghan has failed to elucidate, however, what I consider a very important point, that is, the sampling of the lime. His contention about its deterioration from being exposed to the air in sheds is in my opinion very little based on facts. It may lead to a certain loss, but this I contend would be only superficial in as much as the dusting which takes place in storing the sacks covers the sack itself and prevents almost entirely infiltration of air. Rather would I put before you the best method of grinding, say, ten bags of lime, the quantity mentioned by Mr. Dowling in a previous note, from each truck. This ought to be crushed in a small room with little access of air, so that the floating particles absorb the carbonic acid and moisture from the atmosphere, and so preserve as much as possible the sample to be assayed. This latter should be enclosed in an air-tight jar on the spot, and forwarded to the analyst or assayer. The sample is then thoroughly mixed in the jar, a small quantity taken out and crushed in porcelain with a pestle and mortar till no grits are felt, and then 2 gm. are put into a litre of sugared distilled water. Our custom is to shake this for the best part of an hour that day, allow it to stand over night, and next morning shake for another 10 to 15 minutes. After settling an aliquot part is measured off, filtered and titrated, the first portion filtered through being used to wash out the funnel, receptacle and burette, but not, of course, the titrating vessel. We usually add about ·5 to 1 per cent. to allow for any little absorption of moisture and carbonic acid. In again thanking Mr. Croghan I would express the hope that further discussion will result from the facts and theories he has placed before us.

NOTES ON FEEDERS, WITH A DESCRIPTION OF A NEW DRIVING DEVICE.

By D. J. Pepler (Member).

A high stamp duty is a great factor in the reduction of milling costs, and is moreover responsible for the reduction of costs in all the departments connected with a mine, since costs are usually tabulated on the basis of tonnage crushed. It is essential therefore to obtain as high a stamp duty in the mill as possible, in order to effect a reduction in cost per ton treated. Take for instance a battery of 100 stamps, crushing, say, 600 tons per day with a 400 screen. If the total duty could be brought up to, say, 650 tons per day, the cost of crushing the extra 50 tons would be practically nothing to speak of, as this extra tonnage requires no more labour, power, fuel, etc., than are required to crush 600 tons; the only thing that exception can be taken to being the extra wear and tear on the shoes and dies. In working up the stamp duty from 6 to 6½ tons, it is necessary to see that the stamps are kept plumb and centred, that the dies are placed absolutely dead centre under the stamps, and the mortar boxes re-lined when required, so as to prevent unnecessary space round the dies. It is necessary to regulate every stamp, so that it has its proper height of drop, to be sure that the belts do not slip and that the engine runs at a regular speed. It is highly necessary also that the rock be crushed to a reasonable size in the crusher station. But the greatest factor in high efficiency in the mill is, keeping the feed low and regular so as to enable the stamps to crush everything on the face of the die, for if any accumulation of rock takes place in the mortar boxes, the stamps will be making slimes instead of performing their crushing duty. There is no doubt that judicious feeding plays an important part in the attainment of a high stamp duty, and it is essential that a reliable feeder should be used. To the best of my knowledge, the Challenge feeder is the most suitable known, but the driving device is defective, being cumbersome and too complicated in connection with the suspension gear.

To explain what I mean, the device consists of one disc, one brake wheel, three springs, three pawls, and three shoes with their necessary studs. If one of the three springs happens to break, which occurs frequently, the pawl and shoes fall out, in consequence of which the feeder stops feeding. Again, the pawl and socket in the brake wheel wear pretty quickly, and when the wear reaches $\frac{5}{32}$ in., the pawl and shoe will pass the centre line and fall out at the opposite side. This also causes the feeder to stop. I have had cases where, through the pawl wearing a little short, it has slipped half through on to the central position where it actually jams the disc, causing the feeder to stop feeding.

The above-mentioned defects to the suspension driving gear retard crushing, and are also the cause of 40 per cent. more stems breaking than would be the case if a more reliable driving device were attached to the Challenge feeder. It would not only reduce the breaking of stems by 40 per cent., but would also obviate the breaking of cams and cam shafts to a great extent, and the loosening of heads and shoes. It follows that a considerable saving of time and expense will be effected by a reduction in the above-mentioned breakage and causes of stoppage. When a reliable driving device is attached to the Challenge feeder, by which the feed can be regulated to the exact requirement of the stamps, and which can be depended upon as absolutely reliable, the duty of the mill is increased by fully 8 per cent. Such a reliable device exists, and is called "Pepler's Operative Device to Challenge Feeder."

A few points as to how the grip is produced will, no doubt, be useful and can be followed from the accompanying sketch (Fig. I.).

The feeding stamp striking the rock shaft lifts the lever at A, the lever pressing against wheel at point B on centre line lifts side cheeks at point C, thus causing a twist grip on the wheel flanges at D and E, bringing the wheel forward. After relief of blow on the rock shaft the combination drops back to its original position ready for the next tap. F represents brake block checking wheel from turning backwards.

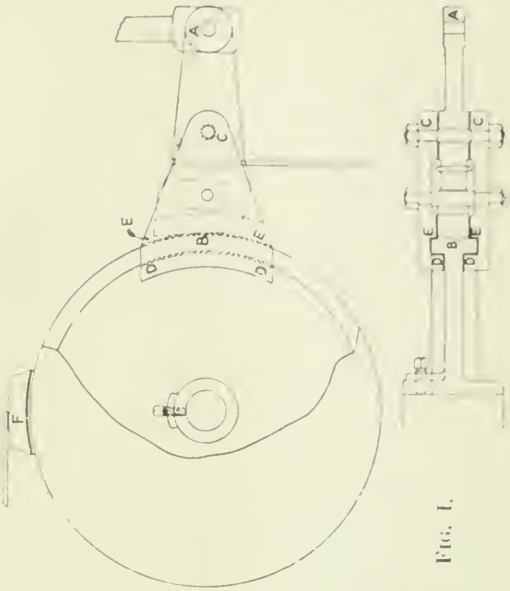

Fig. I.

The combination will move the wheel, which is connected to the pinion shaft driving the feed tray, from $\frac{1}{8}$ in. to any distance that may be necessary, in proportion to the blow delivered by the feeding stamp on the rock shaft. This movement is adjustable, and therefore no more rock is fed into the mortar than the stamps require. Provided that the feed is properly adjusted, the mortar boxes will not run empty and cause unnecessary breakages, the feeder being reliable, feeding the stamps as determined by the adjuster.

I put the first one in on trial, at the Ferreira Deep battery, twenty months ago, and during that time no repairs have been made to the device, it having operated without failure up to the present day. It is in as good working order to-day as when first installed. The device is so sensitive and can be adjusted so closely that the feeder works with each blow of the stamps. I have this device on three-quarters of the feeders in the battery, and the stamp duty has increased considerably in consequence. There has also been a marked reduction in the breakage of stems. A log was kept for comparative purposes when the first device was placed on trial, and was kept for a period of six months to show the difference in stem breakages, if any. I found that 50 per cent. less stems broke in the battery fitted with the new device than in the battery next to it not so fitted, and driven by the ordinary suspension gear. I compute, that over the whole mill the breakages were 40 per cent. less through the introduction of the new device.

It is, I think, proof positive therefore, that a reliable feeder is of great advantage, not only on account of the lesser breakage, but through the higher stamp duty obtained by regular feeding. If any member wishes to see the device at work in order to satisfy himself, he is quite welcome to visit the Ferreira Deep battery, and also the Langlaagte Deep or the Village Deep where there are a few operating.

The President: This is one of those short but instructive papers of which we ought to have more. It rightly attaches weight to simplicity and reliability. They are both very desirable in connection with feeders. The complex feeder is a first class thing for the manufacturer, but it is undoubtedly a worry to the man in the mill, and, therefore, the simple design that Mr. Pepler has put before us commends itself to our attention. I beg to move a vote of thanks to Mr. Pepler.

Mr. C. W. Lee: I beg to second the vote of thanks. At the same time I hope that Mr. Pepler's paper will arouse some of the battery managers, who have been buried in the grave of lethargy for some considerable time. Mr. Pepler's paper has struck a note that will continue beyond this meeting. There are similar devices to that described, which have been in use on the Rand for many years, but owing to the reticence of battery managers they have never been brought to the front. The paper will, I am sure, provoke quite a healthy discussion.

THE SCREEN ASSAY ON THE MEYER AND CHARLTON G. M. UNDER "THE NEW METALLURGY."

(*Read at March Meeting, 1907.*)

By Chris. Toombs (Associate).

REPLY TO DISCUSSION.

Mr. C. Toombs: Since my paper was read, crushing with cyanide solution in the mill has been dropped at the sister plant on the New Goch G. M. Co., and work under ordinary Rand conditions, except for the filter pressing of slimes, resumed, so perhaps a few words about the working of the two plants may be opportune.

On the Meyer and Charlton mine from a few months after the inauguration of the system, more gold was invariably produced than called for by the screen assay, and the surplus was not a matter of a few ounces. This is what led me to experiment with the screen samples, of which this new method for determining the screen assay was the outcome.

On the other hand, from the inauguration of the "New Metallurgy" on the Goch, a serious monthly deficit was almost invariably the rule, and so serious did this become, that after exhaustive experimental work and checking the tonnage, the conclusion was come to that the gold was being stolen, and the whole mine staff fell under suspicion, and two complete staff changes were made inside a very short period. This can hardly be wondered at; here are two mines supposed to be working under exactly similar conditions, one has a monthly surplus of some hundred ounces, and the other a monthly deficit of several hundred. I agree with Mr. Wisdom, that the failure to account for the deficit may have been caused by the "personal factor." With this system very great care throughout the whole plant is required, or very serious losses may occur. On a small compact plant like that of the Charlton, greater care and personal supervision is more possible than on a plant like the Goch, and a striking instance of the "personal factor" is seen in the query put by Mr. Chapman —"What did we do, when the 'hose solution' was higher than the 'tub solution' in gold value?" I never experienced this, but that it was due to careless sampling I have no doubt. What I was struck with on the Meyer and Charlton was the interest taken by the men both in the mill and cyanide plant in their work, and the precautions taken, to prevent any loss of the gold bearing solution, and probably one of the causes of the deficit on the Goch was caused by constant loss of this solution.

Coming to the discussion on the paper:—Mr. Acheson Jones' contribution in the shape of a gold return for the two months, February and March, interested me most, but it would have been a much more finished and generally interesting contribution if he had also given the gold return for two months under the original method of assaying, say, for November and December, 1906, the last two months that that method was in use. I know that at that time Mr. Jones could not so cheerfully say that "the method of determining the screen assay at the Meyer and Charlton gives a result as close to the true value as the system in vogue with the ordinary water process." Certainly to account for the gold to within 0·8 per cent. is extremely good work, and I must congratulate the manager and various heads of departments on the results for these two months.

Messrs. Chapman and Wisdom are both inclined to cavil at my calling the improved method a new one; Mr. Chapman says it was the first method used by him on the Goch, and Mr. Wisdom, who should speak with authority, says it was tried there, because the other method was a failure. I wonder which of these two statements is correct? That assaying the unwashed pulp was tried I quite believe. The method was as follows: Assay pulp unwashed, take another weighed portion and wash with several weighed quantities of water, assaying each to find out the amount of gold washed out, and so by comparing with the "tub solution" assay find out the amount of gold dissolved in drying. This method introduced at least two more assays, and I do not wonder that it was dropped, as Mr. Wisdom says, because it was too troublesome. I distinctly claimed that my method was less troublesome than the old, and that it was quite possible to have the return the same day, and I also claim that my method of calculation is original

Mr. Chapman remarks that there is possibly a loss of gold caused by drying the sample in an iron dish. I do not think there is, as finding one of these dishes scaling, and fearing that the scale might foul the sample, I scraped it off and assayed it, but found no gold. He also says that he had no difficulty in getting beads to agree when assaying the washed pulp. My experience was that only on rare occasions did the beads agree, and when some checks were done by two independent consulting assayers, there was a variation of about 1·5 dwt. between their highest and lowest results on each of six washed pulp samples. As a point of interest I might mention that the average of all six samples returned by one of these assayers agreed very closely with mine.

I agree with Mr. Whitby in what he says about screen samples not being taken often enough, but I do not see how the difficulty he complains of, viz., "unwieldy samples," is to be overcome. I do not know how the sampling is done on the Robinson Deep, but I should say it must be extremely accurate. The 24 hours' sample there for 300 stamps, according to Mr. Whitby, is only 50 lb., sampling being done once an hour, or it is roughly about the same as for the 70 stamps of the Meyer and Charlton, where sampling is only done three times per shift. On working out these figures, I find that the weight of sample per mortar box of 5 stamps per time taken on the Charlton is 6 oz., which does not seem excessive, while the Robinson Deep sample is only ½ oz. I think that members will agree that this is the "smallest quantity compatible."

Mr. White asks, "if the sample is properly taken?" I can assure him that it is, and that the low ratio is not due to loss of solution; and I should say that the loss, if any, would require to be very appreciable, and would be very apparent to the most careless millman, to increase the ratio from 6·5 : 1, as it is approximately at present, to 7·5 : 1, and that would still be a very low ratio of solution to pulp.

Mr. Laschinger's formula is certainly much more concise than the array of figures given by me for calculating the screen assay value in the new method. I tried to arrive at one, but it was complicated, and it did not strike me to simplify it in the neat manner done by Mr. Laschinger.

Mr. Sharwood's contribution is interesting, and he seemed to be on the way to obtain results, when he stopped experimenting. He seems to lay the blame on magnetic iron. I have recently in the course of my work done a good deal of concentrating for pyrites, and I have gone through them with a magnet and separated, and weighed any metallic iron. In no case was it higher than 0·0005 per cent. on ordinary sand samples, and it certainly was not magnetic. I expect the samples he speaks of had been ground on the bucking board, and, of course, this would add to the percentage given above, but I do not see why it should be magnetic, and in any case the percentage would be so small, that on the ordinary fluxing balance it would not be appreciable.

I will now give the results of some experiments which I think show where the trouble lies. A large sample of pulp dried down with gold bearing cyanide solution, was well mixed and divided into two equal parts, one part being well-water washed and re-dried. Assays of both were made, and a grading with assays of the various grades done. The figures are:—

	UNWASHED.		WASHED.	
	Beads	Av'rge	Beads	Av'rge
Original	9·1, 8·6, 8·7	8·80	7·5, 8·2, 8·2	7·97
+60	6·8, 5·1, 6·0	5·97	5·1, 5·6, 5·5	5·40
−60+100	10·9, 8·8, 9·2	9·63	9·0, 8·7, 8·0	8·56
−100+150	12·9, 12·4, 12·8	12·70	12·0, 11·4, 10·6	11 33
−150+200	13·2, 12·6,	12·90	12·8, 11·6,	12·20
−200	9·6, 9·8, 9·6	9·70	7·2, 7·2, 7·2	7·20

It will be seen that there are serious differences in the bead weights both washed and unwashed, except in the −200 portion, in which there is a big drop in gold value on washing, and I should say from the above figures that the very fine gold, which in ordinary samples probably helps to average the coarser particles when assaying, is dissolved, leaving the coarser particles locked up in the coarser grades, and it is not possible to get these evenly distributed throughout the ample taken for assay.

The President: We are indebted to Mr. Toombs for his paper and the very satisfactory discussion it provoked. One might summarise the position by saying that the continuous slimes process has had a set back on the Goch and that its running on the Van Ryn and Meyer and Charlton is being watched with attention.

MINE SUBSIDENCE.

(Read at March Meeting, 1907.)

By Alex. Richardson, A.I.M.M. (Member).

REPLY TO DISCUSSION.

Mr. A. Richardson: The task of replying to the discussion on my paper would have been rendered easier had those who took part in it treated my views in a less complimentary and more stimulating manner. It is, however, gratifying to know that my conclusions have in the main proved acceptable and have been the means of inducing others to bring forward some very instructive matter.

It is interesting to learn that, as Mr. Innes said, our mines owe their coolness to the immunity of the Witwatersrand from any earthquake of recent age, and to this cause might be added the unoxidisable nature of our rock, which, no doubt, greatly assists.

Our freedom from earthquakes is a reassuring circumstance, because mining operations induce a position of unstable equilibrium which must give way to a stable one sooner or later, and earth tremors would hasten the change.

In this connection the following list of earth tremors, notified to our Observatory in recent dates will be of interest :—

Date.	Locality.	Duration in Seconds.	Direction.	Effect.
May 5, 1906	Warmbaths	10	N to S	Shook house
May 20	Warmbaths	20	N to S	Shook house
June 2	Warmbaths	—	—	—
Aug. 27	Alicedale	—	N to S	Shook doors
	Amsterdam			
Dec. 10	Alicedale	4	Various	—
	Holyrood			
	Amsterdam			
	Piet Retief			
	Swaziland			
	Zululand			
	Natal			
March 3, '07	Leydsdorp	—	S to N	Shook house
June 6	Warmbaths	6	—	—
June 15	Warmbaths	6	—	—
June 22	Kasempa Rhodesia	—	ENE to WSW	Shook house

A Milne bracket seismograph properly installed would cost at the outside £80, and considering the importance attached to the instrument in other countries and colonies, Kimberley alone possessing three, the Transvaal might well fall into line, even though seismographic records have no direct bearing on mining. The damp factor, to which Mr. Innes has alluded, appears to depend on the underground water level. This varies greatly in this country and is at present low. It rises with the rainfall, and the accompanying percolation, which latter must be considerable as our spruits and rivers only carry from 2 or 3 per cent., leaving evaporation and absorption to account for the remainder : it is probable that three good wet seasons would make the level as high as it could go. The maximum amount of water in our mines seems to be present at about the beginning of March, and it is about this time, therefore, that the disintegrating effect of moisture should be most noticeable.

Prof. Yates' contribution, with Prof. McNair's address, is a valuable addition to our knowledge of the subject, and they both agree that with deep mines the increased difficulties of roof support will call for drastic changes in our present methods. The subject promises to become increasingly important as the developments in large mines everywhere increase in extent. In the United States extensive caving and crushing of pillars, in some cases reaching over large areas of workings, has taken place during the past year in one or two of the Lake Superior copper mines. One mine, the Atlantic, was forced practically to shut down and for some months has been in process of re-opening. The very deep Quincy mine, also in the same district, has had disturbances of the hanging wall which have called for careful investigation.

Prof. Yates asks the question, "is it worth while keeping the workings of our outcrop mines open in view of the expense that such a procedure would entail ?" I am of opinion that it is. A subsidence is bound to cause grave inconvenience and probably also great expense ; it will bury numbers of valuable pillars which, owing to their scattered positions, it would never afterwards pay to recover ; it will close up hundreds of feet of development which will have to be re-opened in order to make low grade and unworked blocks accessible. It often pays handsomely to re-open an old level for the sake of the pillars along it and the vamping under the rails. How much better would the return be if this were done while the ground was still standing ? Another feature is that it sometimes predisposes portions of the workings in the vicinity to an attack of the same malady through the shaking up of the strata and the consequent increased facilities for percolation. It might be possible, starting at the boundary, to work back along a level taking out everything in the stope above and allowing it to come in. It would be dangerous work because of the strong nature of the hanging which, if it held up over large areas, would cause the resulting crash to be tremendous. The gust of air alone from such a fall would be comparable to an explosive wave.

Estimates as to the depth to which mining can attain are as various as the countries from which they emanate. Prof. Park says:—Mining at great depths will be limited by the underground temperature and the economic conditions. By the adoption of secondary ventilation and methods of refrigeration mining could doubtless be carried on at depths ranging up to 6,500 ft., according to the local increment of temperature.

Mr. A. C. Lane, in an able article in the *Mineral Industry*, gives 10,000 ft. as the general limit, with a possibility of reaching 15,000 ft. under very exceptional circumstances.

If we turn to coal mining we find that the last two Royal Coal Commissions of England have fixed 4,000 ft. as the limit at which coal can be profitably worked.

Geologists have also a word to say in the matter. Spurr states that openings cannot exist beyond depths variously estimated at from 16,000 to 33,000 ft., and Kemp writes as follows:—The maximum depths at which cavities can exist varies from 1,625 ft. for soft shales to 32,500 for firm granites. The maximum depths introduce the zone of flowage, wherein the load is so excessive that the yielding to deformation comes in the way of a viscous flow, or plastic yielding, evidences of which are visible in many gneisses. That such a zone exists will appear to anyone who reflects upon the necessary behaviour and yielding of rocks, which are confined on all sides and yet are compressed beyond their limits of resistance.

The hydraulic filling of workings with culm, sand, sweepings, ashes, and other similar material, to which Mr. Lane Carter refers, is largely practised in the coal mines of Silesia, Westphalia, Hungary, France and America, and a remarkable diminution in the number of accidents from falls of rock as well as other benefits has attended its use.

A great advantage of hydraulic filling is that it fills every inch of the space, whereas careful packing by hand will only fill up 60 per cent. The material packs solid in a very short time and does not run if it is fine; it has been piped underground horizontally for a distance of 3,000 ft. under a head of 656 ft. The water consumption is somewhere about 1½ lb. of water per 1 lb. of culm flushed. Tailings from the cyanide works are being used in Australia; the practice had to be discontinued for a time owing to the noxious fumes overcoming the men, but the installation of a special washing plant overcame the difficulty. Any system of hydraulic filling has not, as far as I am aware, ever been tried in either our coal or gold mines, but it deserves attention in view of its success elsewhere, although the cost of installing and operating would prohibit its use in our deep levels.

Mr. Lane Carter remarks that it is an utter impossibility to keep up a large area of ground when on the move; this is true generally because collapse follows on so very rapidly after the unmistakable signs of subsidence that the danger of men being buried up or imprisoned becomes too great to permit of their entering the affected zone. But there are times when timber stacks filled with waste can be rushed in along the levels and much of the development saved from destruction. Timber alone will not delay the collapse by a single hour, and is only useful where the hanging is bad in small patches and where there is no weight coming on from the strata above. The practice of leaving stope pillars over the hanging wall of an incline shaft sunk in the foot was not a suggestion of mine and meets with my emphatic disapproval, as does also the suicidal practice of stoping under the footwall which has been frequently indulged in in the past.

Facts certainly point to the pillars, as laid down by the present regulation, being in a wrong position; it must be noticed, however, that there were two exceptions to the law of the angle of fracture, and that any change would have an economic, even a legal, bearing. It would be manifestly unfair to deprive an outcrop property of ground which has been taken into its ore reserves in order that some object on the surface of the adjoining deep level, for which ground had already been allocated, should receive support. We may be justified in saying that the present regulation is wrong, but we are scarcely in a position yet to say definitely which other would be right.

To make our mines secure so that posterity may derive benefit from what we leave, as Mr. Wager Bradford suggests, is, I venture to say, a policy that is carried out conscientiously in no mining camp in the world, however self-sacrificing its inhabitants may be in other ways, and however much they may applaud such an admirable sentiment. And when we consider the continual fight which miners wage against natural forces and the uncertainty of how long that struggle can be carried on, it is little to be wondered at that most people prefer to leave for posterity certain sums of money in the bank, if they can, rather than doubtful blocks of ground in the mine.

Mr. Reid Bell's clear description of the settlement of subsided ground, in conjunction with that of Prof. Yates, explains fully what may be expected to occur in any class of formation. He is wrong, however, in the deduction which he drew from my remarks on the Geldenhuis Deep subsidence, as this from its sudden nature was a very likely one to have affected the surface, and not having done so it is regarded by me as one of the strongest arguments in favour of the 1,000 ft. horizon of no surface effects. His remarks on longwall working, of which he has intimate knowledge, are so interesting that I hope he will see his way to extend them for our benefit.

The necessity of a careful examination of wet hanging walls is emphasised in Mr. Whitton's remarks. Now this examination is an excellent safeguard, but in many cases quite ineffective. We have no means yet of testing the solidity of thick slabs of hanging and of thus protecting ourselves against accidents, such as the one which occurred on the New Heriot, where a solid mass, 5 ft. thick and weighing about 400 tons, came away without any warning.

The usual way of testing hanging is by careful examination and by sounding it with a 4 lb. hammer. Now with a 4 lb. a slab 3 ft. thick will not, when struck, emit the hollow sound which indicates looseness, even a 20 lb. hammer will fail to elicit any information if the slab is over 4 ft. thick; a 5 ft. slab cannot be tested at all. It occurred to me that an instrument might be devised which would assist the ear in distinguishing with greater nicety the quality of the sound emitted by different thicknesses of material. Accordingly I made a number of tests with an instrument, a special form of stethoscope, familiar to medical men under the name of phonendoscope. The results have been nugatory so far, and the matter is merely mentioned to draw the attention of those of an inventive turn to a direction where their energies might be well and humanely employed.

The serious problem of falls of rock is solved very considerably by resueing, and from this and other aspects I wish to amplify my original remarks.

It must be borne in mind, that resueing far from being a new thing, is an ancient mining method and that it has been tried in several of our mines and found unsuitable. It cannot be practised where the reef is broken up into stringers, and it has no utility where the reef is wide, its application is also rendered very difficult when bedding planes are not present. A narrow reef with well defined partings is the ideal thing. Its use necessitates a large number of working faces, as it is obvious, that in the alternate breaking of waste and reef the quantity mined of the latter will not be so great as in ordinary working. Furthermore, broken ore cannot be allowed to accumulate in the stopes; this will render underground conditions inelastic. The improvements in ventilation, the lessened danger of accidents from falls of rock, and the increased possibility of making narrow poor reefs pay, are factors which entitle it to respectful consideration.

Mr. S. Richards, the acting manager of the Jumpers G. M. Co., has kindly given me some figures illustrating the two systems, and I have picked out at random a few returns showing the work done by tributors on that mine.

Reef.	Reef Width. Inches.	Milling Width. Inches.	Boys per Shift.	Tons Hoisted.
MRL	6	12	9	50
SR	8	16	20	137
SR	6	12	5	46
MRL	4	8	9	62
MRL	4	12	6	60
MRL	5	10	10	63
MRL	12	15	14	105
MRL	8	16	8	75

In the comparative statement it will be noticed that tributors are charged 10s. per ton on all rock sent out by them. This and the system of paying by the dwt. per ton is an effective check against sending up waste and against losing reef, for it is obviously to the tributors' advantage to send up clean reef and fines so as to get the greatest result, and the obvious rarely requires much pressing when there is money in it.

EXAMPLE OF TRIBUTORS' CONTRACT.

Stope width 30 in. (7½ tons per fathom)
Stope value 4·0 dwt. per ton
Reef width 3 in. (15 cwt. reef per fathom)
Reef value 40 dwt. per ton

Assuming tributor saves 6 in., this would equal 20 dwt. per ton, 1½ tons ore per fathom value 20 dwt. per ton. Value per fathom ...		£6 5 0
1½ tons ore @ 20 dwt. @ 2s. 9d. per dwt. = 55s. per ton Less 10s. per ton reduction charges 1½ tons @ 45s. per ton per fathom	£3 7 6	
Underground Costs—		
1½ tons shovelling, tramming, etc., @ 3s. ...	4 6	
1½ tons reduction charges, etc., @ 10s. 1d. ...	15 1½	
1½ tons residues to dump @ 3s.	4 6	
	£4 11 7½	
Profit per fathom ...	1 13 4½	
	£6 5 0	£6 5 0

Profit per ton milled, 22s. 3d.

STOPING BY CONTRACT @ 60s. PER FATHOM.

Taking the same as above.

Value per fathom ...		£6 5 0
To contractor per fathom	£3 0 0	
7½ tons ore per fathom Costs, shovelling, tramming per ton 3s.	1 2 6	
7½ tons less 20 per cent. sorted at mill 6 tons reduction charge 10s. 1d.	3 0 6	
6 tons residues to dump 3s.	18 0	
	£8 0 0	
Loss per fathom ...	1 15 0	
	£6 5 0	£6 5 0

Loss per ton milled, 5s. 10d.

Mr. Weldon questions the figures given for the crushing strength of pillars and quotes Eytelwein & Euler in support of his argument. Eytelwein's formula, which has been superseded by Euler's, when reduced to its simplest form is to the effect that the strength of rectangular columns varies directly as the longest side multiplied by the shortest side cubed and inversely as the height squared. Similarly Euler's formula is that the strength of a column varies directly as the area squared and inversely as the height squared. These two formulæ are identical when the columns are square. Now Euler's formula can only be used for long pillars, that it is for those that fail by bending. In the case of square pillars of rock Euler's formula cannot be used unless the height of the pillar is 27 times the length of the side. This would mean, taking the ordinary mine pillar of 10 ft. square, that this formula will give quite erroneous results unless the pillar is 270 ft. high.

For shorter pillars, that is those whose height is from about 10 times the side upwards, there are numerous formulæ in use, such as those of Gordon, Grashof, Hodgkinson, Navier, Tetmajer, etc., but these introduce considerations of bending as well as of crushing and cannot therefore be applied. Gordon's, the one in most general use, gives a result which never exceeds the crushing strength of the material. As our mine pillars are neither columns nor pillars, in the strict engineering sense, but short blocks, the only formula applicable is that furnished by the researches of Bauschinger, the eminent German authority on the testing of materials.

There is a general law governing the strength of blocks, such as are used for testing. For test blocks of geometrically similar form the crushing strength varies as the area of the surface on which the crushing pressure acts, or, generally, the strength of geometrically similar test pieces varies as the square of homologous sides. Bauschinger's formula, which holds good to heights up to five times the side, is as follows:—

$$f = \left(\frac{4\sqrt{A}}{C}\right)^{\frac{1}{2}} \left(k + c\frac{\sqrt{A}}{h}\right)$$

A = area of section of prism.

C = circumference or perimeter.

h = height of piece.

f = strength per unit section.

k and c = constants for each material.

It gives at once the law that prisms and cylinders of geometrically similar form have the same strength per unit section, also that in prisms of the same height and sectional area the strength per unit section varies inversely as the square root of the circumference.

For square prisms of different heights the equation becomes $f = k + c\frac{s}{h}$ where s is the side of the square, and h its height.

Bauschinger gives several values for these constants for different materials; the following are for sandstone:—

	k	c
Swiss fine grained sandstone kg. per sq. cm.	283·5	316
Fine grained Bunter sandstone kg. per sq. cm.	327	107·5
Series of tests with square and rectangular prisms of different heights and side ratios; fine grained sandstone	356·5	97

In a series of tests of square prisms of very fine blue-gray Swiss sandstone, the pressure applied normal to bed, and with ratios of side to height of values varying from ·98 to 4·32, the results indicate that the strength increases in the following manner:—

Height = side. Strength per unit section = 1
Height = ½ side. ,, ,, ,, = 1·3
Height = ⅓ side. ,, ,, ,, = 2·5
Height = ¼ side. ,, ,, ,, = 3·4

The average stope pillar on the Rand has probably a height of from ½ to ⅓ of its side or a mean factor, as above, of about two, giving a calculated crushing resistance of 20,000 lbs. per sq. in. on this basis. Now all rock specimens selected for testing are carefully prepared, are of homogeneous material and without flaws. A mine pillar is rarely of homogeneous material, it is certainly not free from flaws naturally, and the vibration of dynamite explosions all round it will have the effect of creating others artificially. It is also subjected to the continued influences of moisture, vibration and uneven pressure: when these injurious factors are given their due weight it will be found, that my estimate of a crushing strength of from 10,000 to 15,000 lb. per sq. in., is, in the present state of our knowledge, quite warranted.

Mr. Weldon remarks that roofs rarely fail by shearing in practice. Failure by bending is certainly the common form of collapse, although from theoretical reasoning the contrary would appear to be the case. Bending is enormously affected by flaws in the rock and shearing very little. A slab will fail by shearing where the greatest weight rests on the smallest area that is at the circumference of the pillar. The area of resistance to shearing is the thickness of the slab multiplied by the circumference of the pillar, and it is only when a flaw follows the contour of the latter that the shearing strength of the slab is greatly reduced. The splitting up of a slab by bedding planes, resulting in separate bodies in course of time owing to moisture, exercises a tremendously prejudicial effect on bending and none at all on shearing, and this one reason alone will explain many instances of roof failure.

The figures relating to the size of pillars, their distance apart, etc., are based on experience, and in this they should carry with them a recommendation to Mr. Coombe.

A few notes on the failure of mine pillars may be of interest. The stress on any oblique plane in a prism loaded with a weight P can be resolved into two components, one a normal component, $P\cos^2 A$, and the other a tangential component, $P\sin A\cos A$ where A is the angle between the normal to the plane and the axis of the pillar. This tangential component or shearing stress becomes greatest when A is 45°, and is equal to $\frac{1}{2} P$. The normal component $P\cos^2 A$ influences the angle and intensity of the stress at which the pillar fails. Acting in a direction normal to the plane of shearing it causes a frictional resistance, which restrains the action of the shearing stress. In very short mine pillars the action along any one oblique plane would be greatly restricted, and this may account for an increase in strength. A cube breaks along symmetrical planes similarly situated with regard to its axis, that is, into six similar and equal pyramids with a rectangular base, and a prism similarly; but in the case of the prism the upper and lower pyramids are terminated by a line instead of a point. A cylinder resolves itself into two cones with the edges split off all round. This action can be often seen in a mine when the pillar is fairly long, pieces flaking off about half way up, the top and bottom remaining solid, until at last it degenerates into a wasp-waisted affair of shattered rock.

Mr. Boright's remark that falls are more prevalent before the month of May supports the supposition that percolation resulting from the rainfall exercises a marked effect. The danger consequent upon the breaking of a footwall in order to obtain waste for packing in steep stopes, to which Mr. Homersham alludes, does not, of course, apply to those of moderate dip. The main point to be considered then would be the thicknesses of hanging and foot, supposing the waste to be required in a stope occupying an intermediate position.

I regret that the scope of my paper will not permit me to discuss fully with Mr. Coombe the relative merits of practice and theory, so will conclude by observing that a wrong idea, even if embodied in a formula, is not such a bad thing, as it may be confuted. It is the want of ideas that is anathema.

The President: Mr. Richardson has certainly struck a field which is quite new to us, though I am sure that as years go on we shall hear more and more of it, because we must expect something like a general subsidence right along the reef. He mentions the matter of a seismograph. Mr. Innes, the Director of the Observatory, also touched upon that subject in his contribution to the discussion, and if I remember rightly you listened to his suggestion that the Observatory should have a seismograph very sympathetically. I had no idea that these instruments were so cheap. A matter of £80 is not much, considering the utility of such an instrument, and I would like your views as to whether the Society should move in the matter of seeing that the Observatory is provided with one.

It was agreed that this matter should be left in the hands of the Council.

The President: May I make a remark in connection with the Jumpers mine, which Mr. Richardson mentioned? I was round there to-day with some of my students, and I hope that before long the Council will be able to arrange a visit for the members of the Society. Undoubtedly it is the most interesting mine on the Rand. You not only have resueing there but you have special forms of timbering and supports which you see nowhere else on the Rand.

NOTES ON SOME RECENT IMPROVEMENTS IN TUBE MILL PRACTICE.

(*Read at April Meeting, 1907.*)

By Kenneth L. Graham, M.I.M.M. (Member).

DISCUSSION.

Mr. J. Kennedy: In spite of having used banket exclusively as a substitute for imported pebbles, for the past 15 months, I have always been under the impression that better results could be obtained by using pebbles. I fail to see how an irregularly shaped piece of ore can possibly be as effective as a smooth round pebble.

Without presuming too much, I venture to suggest that, in Mr. K. L. Graham's test, the No. 2 mill consuming pebbles was underloaded, as I do not consider 670 lb. sufficient feed for a mill of this size per 24 hours. It is not surprising that Mr. Fraser Alexander got better results with banket in his test. As stated, his trial was of shorter duration, and two tons of Danish pebbles were fed daily. If not an impertinent question, I should like to ask Mr. Alexander, for what period was the feed of two tons per mill per 24 hours continued?

With Mr. Graham's theory of the higher percentage of work being done by impact, rather than by grinding, I entirely disagree, and for this reason:—I had come to the conclusion some time back that, manganese liners had not been given a fair trial before being condemned, as, at the Glen Deep, the life of manganese liners was about 90 days, while at the Robinson Deep, the liners lasted only about 50 days; I, therefore, set to work to ascertain the cause of this marked difference in life, and in endeavouring to approach as nearly as possible the real thing, *i.e.*, manganese lining, I found that by loading a glass jar with small pebbles, which had been discharged from the tube mills, the pebbles reaching just under the neck of the jar, when lying on four rollers, and clean sand and water at 1 to 1 being added to reach just under the centre of the jar, the jar then being made to revolve—the result was just as I had anticipated. Instead of the load being raised sufficiently high to enable the pebbles to roll and tumble over one another, the load, with the exception of a few pebbles slipped, on the side of the jar whilst revolving.

More pebbles were then added to bring the load well up over the centre, and the jar was started again. This time, much to my satisfaction, the pebbles, with pulp clinging to them, began rolling and tumbling over one another dawn the inclined surface of the load, formed by the rotary motion of the jar.

It is interesting to note how the pebbles and pulp rise at the ascending side of the jar, although, to all appearances, the water is level to about the outflow on the descending side.

Some members might still be of the opinion that this experiment does not prove that the grinding is not done by impact, and for their benefit I should like to explain that, in this experiment, when the jar was under loaded, there was scarcely any sound noticeable while the jar was revolving, but, when the pebble load was increased to the point mentioned, there was the noise of the pebbles striking the liner, which coincides with Mr. S. H. Pearce's experience while the manganese liners were being tried at the Glen Deep, and which was mentioned while Mr. H. A. White's paper on "The Theory of the Tube Mill" was being discussed before this Society about two and a half years ago. Mr. Pearce stated that a good deal of difference was noticed in the sound of the tube mill when this liner was first run, inasmuch as there was an absence of rumbling which had been a feature previously. He also stated that there was a tendency for the pebbles to wear flat, and the crushing efficiency dropped considerably. Later, the rumbling occurred again, and the crushing efficiency increased. He also concluded by remarking, that he supposed that there was considerable slip on the smooth surface when starting, and that later, the liner had acquired a rough surface and raised the pebbles a little higher than the smooth surface had done, which fact causes crushing to be accomplished instead of reduction by attrition. The manganese liners when discarded at the Robinson Deep, were as smooth as glass, and this proved that Mr. Pearce's supposition was incorrect. The sole cause of the absence of rumbling, and the dropping of efficient grinding was, that the mill was under loaded. An important point which appears to have been overlooked, when advocating the theory of crushing by impact, is that, as soon as the rolling or falling pebble touches the water, the pulp clinging to it becomes disengaged, and the bare pebble strikes the liner, and this accounts for the rumbling referred to.

With reference to hydraulic classification, I run seventeen classifiers, five abreast in the first row, and six in the two following, and use water on all, finding it essential to do so when dealing with a big tonnage with a limited tube mill installation.

The longest life of silex liners, 4 in. thick, I have experienced is, 108 days, and a lining of local chert, 4½ in. thick, has been running for 102 days, and to all appearances will last a fortnight longer; but it seems that the days of silex and cherts as liners, are numbered, judging by the success attending the lining accidentally invented at the Los Estrellas mine, El Oro, Mexico,* where the inventor had lined his mill with riffle-like castings in order to raise the pebbles nearly to the top of the mill, thus increasing the height of drop. When the mill was opened after several hours' run, it was found that the pebbles had become wedged into the riffles, and thus formed a lining of themselves. After eight months' run, the wear of the casting was found to be so slight as to indicate a life of two to three years.

A FEW NOTES ON THE REFINING OF BASE BULLION.

(*Read at May Meeting, 1907.*)

By C. W. Lee and W. O. Brunton (Members).

DISCUSSION.

The President: I think Mr. Lee has an apparatus which he would like to show the members.

Mr. C. W. Lee exhibited an air pump for generating compressed air. He pointed out that the air from the compressors could not be made available owing to the moisture contained therein, apart from the heavy cost of conveying as most of the smelting works are situated some considerable distance from any air connections, and therefore some local means of generating compressed air cheaply must be adopted.

The apparatus shown was very simple but efficient, consisting merely of a petrol tin reservoir and a motor pump which could be worked by a native.

Mr. T. Lane Carter: Since reading the paper I have been struck with the trouble our friends have had with their cyanide bullion. It is apparent that there is a difference in the fineness of gold along the reef in the banket. I have worked out the average fineness from our cyanide works for the last three months, and find it to be about 892 fine. We have had one or two bars as high as 906, and several bars over 900, but the average is 892. That shows how fine the gold is on the West Rand. We have also been on the look out for copper and other metals in the ore, but my experience out west is that we have extremely small quantities of base metals in the banket. We have succeeded in finding traces only of copper, zinc and lead. I have been wondering whether the trouble Messrs. Lee and Brunton have had, is due to the base metals being in the banket in larger quantities in East Central section of the Rand.

Prof. J. A. Wilkinson: I should like in the first place to congratulate the authors on having successfully introduced a practical method of refining poor cyanide bullion by means of air, but I was sorry to find on reading their paper that data, such as are required for the study of the scientific aspect of the problem, are entirely wanting. It was from this point of view that my attention was drawn to the paper as I was studying the Bessemer steel process at the time. I would, therefore, ask the authors if they could supply the missing data, so as to make it possible to work out the problem.

The President: We will not close this discussion to-night, but keep it open until the next meeting.

THE UTILISATION OF WASTE HEAT IN SLIMES SETTLEMENT.

(*Read at June Meeting, 1907*).

By A. Salkinson (Member).

DISCUSSION.

Mr. W. R. Dowling: Throughout the pages of our *Journal*, and other technical papers and books, are to be found scattered references to heating cyanide solutions, but really no definite information or figures are given so far as I have read.

I do not think there is any doubt that heat does accelerate the settlement of ore slimes. Most practical cyanide workers are satisfied that the settlement is better during the summer months than the winter. Mr. Salkinson is the first on these fields to apply the artificial heating of the solution on the working scale, and to produce results which prove the value of the scheme beyond a doubt. His figures go to show that by increasing the slime solution temperature to about 80° F., the capacity of the plant is increased by about 20 per cent. If heat had to be produced specially for the slime works, there would be room for doubt as to whether it paid, but, as Mr. Salkinson points out, there is much waste heat on the mines which can be used cheaply. I think our mechanical engineers now have a good opportunity of developing the scheme of condensing steam with the cyanide solution, which the cyanide works would, no doubt, lend them, on condition that it was returned so many degrees hotter. What surprises me is, that our mechanical engineer members have not jumped at the idea. In using warm water for milling, as far as the cyanide works are concerned, some decomposition of the sulphur compounds of the ore might take place, but this would probably not be more than could be easily overcome. For amalgamation, I think, most mill men will agree, that no harm is done as long as the temperature is constant, and, of course, not too high —say, about 80° F. I am anxious to have Mr. Salkinson's scheme tried on the plant in which I am interested, and hope later to report that it has been introduced.

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* See this *Journal*, p. 60.

NOTES ON WAIHI ORE TREATMENT.

(Read at July Meeting, 1907.)

By Ralph Stokes (Member).

DISCUSSION.

Mr. W. R. Dowling: I have read Mr. Stokes' paper with interest, as it is a description of the practice of other countries by one of ourselves. Tube milling in Australia is so different to ours that it is very difficult to make comparisons. The economy of the honeycomb liners (Barry's patent), would probably depend on the existence of a cheap and suitable, local material for filling. If it had to be imported or transported any distance, the only saving, if any, would be the dressing of blocks, against which would have to be set the cost of the iron castings, and the labour and cement for filling. The castings, I take it, wear out with the filling. I think the economy, or otherwise, of this ingenious idea would have to be proved by actual experiment, using some local material, of which there is plenty in the Transvaal. I believe some of our tube milling men are making blocks of worn out liners and small pebbles. The procedure, as I saw it on the Glen Deep, in the hands of Mr. Osborn, the cyanide manager, was to break up the silex and pebbles into small pieces, and fill them with cement into wooden moulds. When set, the blocks were removed from the moulds. The moulds are shaped to produce a block, curved at the back to fit the circle of the mill, and with sufficient batter. The honeycomb liners certainly have the advantage of being built in very quickly, reducing the time of stoppage to even less than the creditable records of some of our local men.

A separate house for the amalgamating plates is a point that recommends itself. The hanging up of the mill, because the gold recovery portion of the works require attention, seems to me wrong in principle. With a plate to each 5 stamps, the stoppage of our batteries for dressing and scraping plates, amounts to about half-an-hour per day, or about 2 per cent. of the possible time. Mills as constructed on these fields require men capable of running machinery as well as recovering gold by amalgamation, qualifications which are quite opposite. The better arrangement is that described by Mr. Stokes where an engineer attends to the machinery and a metallurgical man to the gold recovery, in a separate building. Finally, as pointed out by Mr. Stokes, the danger of theft, if it does exist, is very much reduced.

With reference to the treatment of slimes, I think that Mr. Stokes' prophesy—that filters will take the place of presses—is true, in localities where the decantation process does not prove better than either.

Mr. C. E. Deakin (contributed): I read Mr. Stokes' paper with very great interest, as for some short time I was employed on the Waihi Company's property. The first part of the paper is confined to methods of treatment which have been introduced since I left there, and it is more particularly as regards the milling practice that I venture these few remarks.

In dealing with the milling arrangements of the three mills belonging to the Waihi Company it must be borne in mind that the two principal mills, the Waihi (90 stamps) and the Victoria mill at Waikino (200 stamps), the latter situated some 3 or 4 miles further down the Ohinemuri River, were originally installed and equipped for dry-crushing with stamps, the ore having first been dried in big kilns cut out of the hill side. It was only when the great advantages of wet-crushing could no longer be argued against, and the quantity of sulphide ore, which necessitated concentration, had increased to an exent beyond the capacity of the small Union mill (40 stamps) designed to cope with it, that the company converted their mills to wet-crushing plants.

Mr. Stokes states in his paper that the practice of keeping the amalgamation entirely distinct from the crushing department is peculiar to the Waihi Company, at least in Australasia. Perhaps it may be questioned if the Waihi people would have adopted that system, had not as I have before said, the two principal mills been built for dry-crushing, and consequently plate room was not taken into account: while in the Union mill the neighbourhood of the stamps was so dark that careful amalgamation would have been impossible except through the aid of artificial light. The advantages of the arrangement are many, for example, the entire absence of grease, the full efficiency of the plates and the doing away with the necessity of hanging up any stamps whilst the plates are being examined and periodically dressed and cleaned up.

While not maintaining that the Rand companies should alter their plants already built, though a good case could be made out even for that, I think it can be proved that it would be economical for future mills to be built on that principle. For purpose of calculation I have taken a 200 stamp mill, expecting few, if any, of the deep deeps will be worked on a smaller basis. A conservative estimate of half an hour stoppage per plate per day, would represent on the stamping capacity above mentioned and taking a 5-ton stamp duty, a clear gain of 20·8 tons crushed per day, which allowing for an extraction by amalgamation of 4 dwt. per ton would equal 4·16 oz. of fine gold, value £16 13s. 4d. It would also be the means of slightly increasing the white labour on these fields as the extra labour required should only absorb £6, taking three shifts consisting of one amalgamator at 25s. per shift, and assistant at 15s. Against this £6 a saving might be effected by decreasing the number of hands required in what would be simply the stamping department. For the purpose of these few notes I have not taken into consideration what would probably take place, namely, an increased plate extraction, since the plates would have the undivided attention of the amalgamators.

I would like to mention here that, in at least one of their mills the Waihi Company use Muntz metal plates in preference to copper. In that particular mill they were treating the larger quantity of the sulphide ore then being obtained from the mine, and it was considered, and with good reason, too, that the Muntz metal plates were easier kept clean and in a bright condition, due possibly to the slight galvanic action set up by the zinc-copper couple. The Muntz metal, not absorbing the mercury like copper, requires rather closer attention and more frequent removal of the amalgam.

The plates were usually thoroughly cleaned up each month, and about once every two months steam was employed to remove as much amalgam as possible, for which purpose a light canvas frame was placed over the plate and steam introduced at one corner through a ½ in. hose and exhausting at the other corner diagonally opposite. It was usually found that, if due care had been exercised the plates would resume their normal condition within 24 hours of the operation being concluded. A weak solution of H_2SO_4 was used for dressing the plates.

A further point to be noticed in Mr. Stokes' paper is the fact that all amalgamation is done after the tube mills and not as the Rand practice usually is before and after the re-grinding process. I consider that the amalgamating should take place when the ore is at its finest, otherwise I am inclined to think that a certain amount of scouring is inevitable in view of the coarse screens now employed when tube mills are also in use. The present practice has the further disadvantage of having two amalgamating departments to control instead of one.

In concluding, I would like to state that the remark about the residues having already proved the snare and delusion of one adventurous syndicate is hardly fair to the business acumen of the men connected with it. I think I know the plant to which the writer of the paper refers, and it was erected with the idea of treating, not the current residues of the Waihi mill, but the old accumulation of tailings from the early days of the mine, before the cyanide process came into general use, when the course of treatment consisted of dry crushing with stamps and subsequent pan amalgamation with an extraction varying from 40 to 60 per cent. These old tailings have been undergoing a process of concentration due to the flow of the river for the past 10 or 12 years. If the syndicate had erected the plant with the intention of treating the current residues from the Waihi Company's mills they would not have placed it some considerable distance up the Ohinemuri River from the Waikino mill (200 stamps), seeing that all residues are discharged into the river.

The meeting then closed.

Contributions and Correspondence.

NOTES ON ROUTINE MINE SAMPLE ASSAYING.

Some eighteen months ago I devised a scheme whereby the green copper cupel stain could be utilised to obviate to a great extent the interchange of mine samples during the several processes undergone in assaying.

It was found that, by placing 25 mgm. of c.p. copper foil in a given fluxed sample before smelting in the wind furnace, a distinct green stain was noticeable on the corresponding cupel without practical detriment to the accuracy of either the bullion or fine gold determination of that sample, ordinary care in the manipulation of muffle temperatures having been observed, and magnesia cupels used.

We run our mine samples in lots of 28, in both wind and muffle furnaces, seven rows of four. We have four cupel trays which were made on the property, each tray having 28 partitions in seven rows of four. We also have sample trays on the same lines, but sufficiently large to allow of the fluxed samples being placed in them ready for carrying to the wind furnace. Cupel and sample trays are conspicuously marked I., II., III., and IV. respectively.

The fluxed samples are placed on the sample tray as under :—

1	2	3	4
5	6	7	8
9	10	11	12
13	14	15	16
17	18	19	20
21	22	23	24
25	26	27	28

No. I. Tray.

All bags are closed with the exception of 1, 5, 9, 13, 17, 21, and 25, and into each of these seven open bags is dropped a piece of c.p. copper foil, weighing approximately 25 mgm.

No. II. Tray.

2, 6, 10, 14, 18, 22, and 26 are coppered.

No. III. Tray.

3, 7, 11, 15, 19, 23, and 27 are coppered.

No. IV. Tray.

4, 8, 12, 16, 20, 24, and 28 are coppered.

So that when the four cupel trays are placed side by side with cupels in them after cupellation, their appearance may be described as under :—

I.	II.	III.	IV.
O O O O	O **O** O O	O O **O** O	O O O **O**
O O O O	O **O** O O	O O **O** O	O O O **O**
O O O O	O **O** O O	O O **O** O	O O O **O**
O O O O	O **O** O O	O O **O** O	O O O **O**
O O O O	O **O** O O	O O **O** O	O O O **O**
O O O O	O **O** O O	O O **O** O	O O O **O**
O O O O	O **O** O O	O O **O** O	O O O **O**

It will be seen that stained cupels are represented in thicker type than those not coppered.

An experiment was made to determine the cupellation loss when varying weights of gold bullion (10 per cent. silver), 20 gm. lead foil, and 25 mgm. c.p. copper foil were taken. This furnishes a charge practically identical with the average coppered lead button obtained by us in the assaying of mine samples.

Now, against this surcharge I found 4·5 dwt. silver (weighed collectively) in 400 gm. lead foil. In order to obtain these figures I was obliged to run some 35 cupellations with 25 mgm. copper foil (and under the same conditions as in tabulated experiment above) before I could get 20 prills together, owing to the smallness of these prills which in many cases went into the cupel. Taking into account this 4·5 dwt. silver, the gain is converted into a loss of ·5 dwt. on 198·5 dwt., *i.e.*, approximately ·25 per cent., and as only one sample in four is coppered, the average loss through copper dwindles to the negligible percentage of approximately ·06.

1 oz. = 1 mgm., weighed to ·5 dwt

No.	Rough Weight of Bullion before Cupellation.	Actual Weight of Bullion Collectively before Cupellation	Collective Weight of Prills after Cupellation.	Gain.
	Dwt.	Dwt.	Dwt.	Dwt.
1	3·5	15·5	16·5	1·0
2	3·5			
3	3·5			
4	1·5			
5	4·5			
6	6·0	30·5	31·5	1·0
7	6·0			
8	4·0			
9	6·0			
10	6·0			
11	10·5	51·0	52·0	1·0
12	10·0			
13	9·5			
14	11·5			
15	10·0			
16	20·0	101·5	102·5	1·0
17	19·0			
18	21·5			
19	20·0			
20	20·0			
Totals ...		198·5	202·5	4·0

The main point claimed for the copper stain check system is, that few mistakes occur when assistants know that blunders are apt to advertise themselves, and further, that interchange of trays is rendered obvious.

Implicit confidence in one's own work may be good, but unnecessary trust in others is bad.

H. Monckton.

Village Deep, Ltd.,
July, 1907.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Analysis of Wolfram and Hubnerite.—"The coarsely powdered mineral is heated with three times its weight of sodium hydroxide in a silver crucible over a spirit lamp. Half the alkali is first melted and allowed to cool. The substance is then added, and afterwards the remaining alkali. Fifteen seconds suffice to complete the reaction, and the mass is then taken up with water and filtered. The alkaline liquid is oxidised by bromine and acidified with nitric acid. Sulphur is then determined by adding barium nitrate, the excess of barium is removed by sulphuric acid, and the acid liquid evaporated to dryness to remove most of the silica. Arsenic, molybdenum, and tin are next removed by sulphuretted hydrogen. The liquid is rendered ammoniacal, again treated with sulphuretted hydrogen, and the tungsten precipitated as sulphide by cautious addition of hydrochloric acid, the precipitate being calcined and weighed. Iron and phosphorus are determined in the filtrate. The residue remaining unattacked by the sodium hydroxide is readily decomposed by hydrochloric acid. It contains iron, manganese, cassiterite, and gangue (quartz), but never tungsten. The freedom of the barium sulphate precipitates from silica, and tungstic, columbic, and tantalic acids must be ascertained, as also that of the silica from the acids and that of the tungstic anhydride from silica."—P. NICOLARDOT, Compt. rend., 1907, *144*, 859—861.—*Journal of the Society of Chemical Industry*, May 31, p. 530. (A. W.)

METHOD OF WASHING LEAD SULPHATE PRECIPITATES.—"When lead is determined as sulphate, the precipitate may be washed with a solution containing 0·7 per cent. of ammonium sulphate, instead of using alcohol, as is customary, or dilute sulphuric acid followed by alcohol. The small quantity of ammonium sulphate retained, in no way interferes with the subsequent incineration of the filter paper."—L. L. DE KONINCK, Bull. Soc. Chim. Belg., 1907, *21*, 141—149. —*Journal of the Society of Chemical Industry*, May 31, 1907, p. 555. (A. W.)

TINOL, A NEW SOLDER.—"The author reports on a new solder, tinol, its conductivity and strength. Tinol consists of a soft solder metal, containing tin and lead in varying proportions. This composition is disintegrated to a very fine powder, and mixed with dry powder soldering fluxes. It is then worked up into a paste or viscous fluid with glycerine, alcohol, or other such media. This paste was spread over the test joints, and the joints made either by simply warming or by passing over them hot soldering irons. It was found possible to join all sorts of junctions, which otherwise would have had to be made by other methods, such as screw-clamps, etc. The conductivity experiments show a 4 to 5 per cent. higher value for tinol, while a strength test gave similar results to the older methods of soldering."—M. CORSEPIUS, Verhandl. des Ver. zur Beförd. des Gewerbfleisses, Berlin, 1906, 237—244: Proc. Inst. Civil Eng., 1906-7, *167*, 72.—*Journal of the Society of Chemical Industry*, May 15, 1907, p. 472. (A. W.)

QUANTITATIVE DETERMINATION OF FLUORINE.—"The author has obtained satisfactory results in the determination of fluorine in fluorides by a method devised by W. Lenz. The substance is weighed into a small platinum crucible and mixed therein with a small quantity of pure calcium oxide. The remainder of the crucible is then filled with calcium oxide, and the whole is inverted in a larger platinum crucible, the annular space between the two being also filled with calcium oxide. The whole is then heated gradually to incipient red heat, cooled, the contents of the crucibles cautiously slaked with water, and dilute acetic acid added so long as evolution of gas occurs. The liquid is now treated with one-tenth of its volume of alcohol, and after several hours the calcium fluoride is filtered off and washed with dilute acetic acid containing some alcohol, until the washings no longer give an immediate precipitate with ammonium oxalate. The calcium fluoride is then dried and weighed, and may afterwards be converted into calcium sulphate and again weighed. Check experiments with pure sodium fluoride and ammonium fluoride showed that the method uniformly gives results about 0·8 per cent. too low, which, considering the difficulty of fluorine determinations, represents but a comparatively small error. Commercial ammonium fluoride was found to consist of the normal fluoride, NH_4F, and not, as is frequently assumed, the acid salt, NH_4FFH. The author has also examined the iron fluoride obtained by dissolving metallic iron or ferric oxide or hydroxide in hydrofluoric acid in presence of nitric acid, and finds that instead of being ferric fluoride, $Fe_2F_6,9H_2O$, as is usually stated, it is a ferro-ferric compound of the composition $(FeF_3)_2,FeF_2,10H_2O$." — E. DEUSSEN, Monatsh. Chem., 1907, *28*, 163—172 —*Journal of the Society of Chemical Industry*, May 15, 1907, p. 490. (A. W.)

DETERMINATION OF TIN AND TUNGSTEN.—"The mixed tungstic and stannic oxides are heated in a covered porcelain crucible with zinc dust or zinc filings, for a quarter of an hour, and after cooling, the mass is heated with 1 : 2 hydrochloric acid till no more hydrogen is evolved, and all the metallic tin is, therefore, dissolved. To the cooled liquid, potassium chlorate is cautiously added till the blue colour has entirely disappeared, and the tungsten is entirely in the form of dioxide.* The liquid is now diluted with at least 1·5 times its volume of water, and allowed to stand for 24 hours, when the whole of the tungstic oxide is deposited. It is filtered off, washed first with water containing nitric acid, and finally with a hot dilute solution of ammonium nitrate, dried, ignited, and weighed. In the filtrate the tin is precipitated as sulphide."—E. DONATH.—*Engineering and Mining Journal* from *Zeit. f. angew. Chem.*, 1906, vol. xix., March 23, 1907, p. 573. (A. McA. J.)

SOME EXPERIENCES IN ASSAYING TELLURIDE ORES.—"It has long seemed to the writer that too little is known regarding the losses in the assay of telluride ores, and while the following notes may cover ground that has been more thoroughly gone over by other assayers who may have worked out a method of testing to their satisfaction, yet he feels justified in publishing them for several reasons.

Ore No. 1.—This sample was brought to the writer some seven years ago, with instructions to assay for gold. The ore was of a lead-grey colour, with metallic lustre, and free from quartz or gangue of any kind. Tellurium was at once suspected, and its presence was determined by the sulphuric acid test. It was also noted that sulphur was present, and a little later bismuth was detected.

The sample was prepared as usual, and charged into crucibles with a heavy litharge flux, such as was in common use on Cripple Creek ores. Owing to the small amount of pulp at hand, only ¼ assay ton charges were used. Silver was added. Resulting lead button was slightly brittle, but not enough so to be considered detrimental. On cupelling no button was found. Another charge was made up, more litharge and a larger piece of silver foil added. Still no button. The cupel was then ground up and assayed. Result, no button. The cupel was run again and this time a very small button was obtained.

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* ? trioxide. [Ed. Com.]

Prof. Furman was greatly interested, and advised a combination method, consisting of treatment with nitric acid to oxidise the tellurium, precipitation of the silver with sodium chloride, and scorification of the residue for gold.

This was followed out with the astonishing result of some 50 oz. of gold and 3,000 oz. of silver per ton. The button was found to be pure silver. The sample was then submitted by Prof. Furman to a mineralogist of Denver, who classified it as an un-named mineral—a sulpho-tellurate* of bismuth. This gentleman stated that he had recently had a sample of the ore, and advised the same treatment for it as already used, adding that the precaution must be taken to precipitate the silver with salt and not hydrochloric acid, as otherwise a combination would take place between the acid and the tellurium and re-dissolve the silver. Owing to lack of ore no further tests were made.

Ore No. 2.—A little later in the same year, a control sample was received from Cripple Creek with instructions to assay for gold. The sample was pulped when received, but tellurium was suspected and litharge charge run. Result, no button. Ore was tested and selenium, as well as tellurium, was found. Remembering the former experience, the ore was treated with nitric acid as before, and residue scorified with addition of silver. A gold button amounting to about 7½ oz. per ton was recovered.

The sender of the ore was notified as to its character and method of testing with instructions to tell his assayer how the ore was run 'in case he found no gold.' Much to the surprise of the writer, he received a polite letter from the assayer against whom the work was done (a man of wide experience) requesting that the assay be repeated, as he had obtained results of over 100 oz. per ton. He kindly explained his method of assay, which consisted in scorifying 1-20 assay ton of ore with an excess of lead, re-scorifying the lead button twice, and then cupelling.

This was done and his results checked (and this in direct contradiction to the published statements of at least two authorities on assaying that scorification is not applicable to tellurium ores). Considerable correspondence with the Cripple Creek assayer brought out the fact that ore from this particular mine always gave this trouble, while other ores of the district yielded their values to the ordinary litharge flux. The writer brought up the point that in all probability the acid treatment had resulted in some combination with the selenium by which the gold in the ore went into solution—the filtrate not having been tested for gold. The other party inclined to the opinion that the gold was probably rendered so fine by treatment with the acid that it passed the pores of the filter paper. At any rate, the acid method did not seem to work on this particular ore.

Having considerable pulp for experimental purposes, tests were made to determine the exact amount of gold in the ore by dissolving and precipitating. The result was 156 oz., or 50 per cent. more than what was obtained by the method used.

At the suggestion of a well-known chemist, 1-20 assay ton of the ore was wrapped in tissue paper and dropped into a scorifier containing lead that had started to scorify. The result was about 140 oz., or close to the actual value of the ore. This would seem to indicate that some loss was due to the volatilisation. No further tests were made.

Ore No. 3.—This was a sample of float from a grazing section of Colorado, and was not suspected to contain any values in gold or silver. Testing it showed a very large amount of tellurium. The gangae was clay, or some variety of shale. The ore was run with the heavy litharge flux recommended by Hillebrand and Allen, and gave results of 8·80 oz. gold and 58·80 oz. of silver per ton.

One-tenth of an assay ton was scorified with an excess of lead and the resulting lead button was re-scorified and cupelled. Result, 20 oz. gold and 121·60 oz. siver per ton.

The acid treatment was used on another sample of the pulp, and gave results of 20 oz. gold and 109 oz. silver per ton. No further tests were made on account of lack of ore. The scorification method seemed to be the best for this ore.

Ore No. 4.—This ore was very similar in character to No. 1, although run several years later. A large sample was secured and the tests made by different methods, with the idea of obtaining the best one. One-tenth of an assay ton was run with the heavy litharge charge. The lead button was malleable. Result, 3·20 oz. gold and 1,534·00 oz. silver per ton. The silver button was dull in colour, and the cupel showed the tellurium stain with some minute beads of silver around the principal bead.

The resulting silver button was dissolved, and the silver precipitated with salt. Silver recovered weighed at the rate of 1,180 oz. per ton, against 1,534 oz.; showing that the button had consisted of something besides the silver. The filtrate was tested and showed bismuth in large quantity.

The cupel from the first button was ground up and tested. Litharge flux was used. Lead button was malleable, but resulting silver button was flat and dull in colour. Cupel still showed minute beads of silver and tellurium stain. Results from cupel, 6·40 oz. gold and 4,907 oz. silver per ton. Dissolved silver as before and precipitated. Resulting silver button weighed at the rate of 3,985·20 oz. per ton. Filtrate again showed bismuth in large quantity, and tests proved it to be bismuth beyond all doubt.

The second cupel was run again as before. Result, 412 oz silver and 0·40 oz. gold per ton. The button of silver was round and bright, and proved to be pure silver. This cupel was again run. Result, 40 40 oz. silver and trace of gold per ton. The silver button was pure.

Total recovery by the treatment was 10 oz. gold and 5,617·60 oz. of pure silver per ton.

One-half assay ton of the same sample was treated with 100 c.c. of water and 50 c.c. of nitric acid. Allowed to stand in warm place until action ceased, 50 c.c. more nitric acid added, and after all action ceased, diluted with 500 c.c. water. Filtered and washed. The silver was precipitated with salt and lead acetate, and sulphuric acid added. Collected on filter, washed, dried and scorified. Residue was scorified for the gold. Both lead buttons were malleable, and the results were, 14·32 oz. gold and 6,465 oz. silver per ton.

One-tenth of an assay ton and 0·05 assay ton of the ore were then scorified with an excess of lead and the resulting lead buttons re-scorified with more lead until the final lead buttons were hard but not brittle. Results from 0·1 charge, 7,042 oz. silver. From 0·05 charge, 7,354 oz. silver. The gold values were neglected. The silver buttons were flat and dull in colour. They were dissolved and re-precipitated with salt, and weighed as follows: One-tenth. 5,664 oz.; 0·05. 6,230·40 oz. per ton. The filtrate showed bismuth on testing.

Results of all tests were by using litharge flux and crucible assay, re-running cupels three times and obtaining the pure silver by dissolving and precipitating: 10 oz. gold and 5,617·60 oz. silver per ton. By the acid treatment and precipitation of the silver, 14·32 oz gold and 6,465 oz. silver per ton. By scorification, taking special pains to get rid of the tellurium by repeated scorification, 5,664 oz. silver and 6,230·40 oz. silver per ton.

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* ? sulpho-telluride. [Ed. Com.]

Conclusions.—From these tests and others, the writer is inclined to believe that all telluride ores will not yield their values to the same treatment. A straight telluride ore, even of high percentage, will apparently give good results by the scorification method, but where other metals, such as bismuth or selenium are present, it fails to work well. The crucible method with excess of litharge has not proved to be very good in the writer's case, although other assayers seem to get satisfactory results. Whether they have had the same ores to contend with is not known to the writer. The combination method should yield good results if properly carried out.

The principal losses seem to be in absorption by the cupel and not by volatilisation. This will depend somewhat on the ores, however. The presence of other metals than gold and silver in the button in the cupel is something that was new to the writer, and not generally known. He has thoroughly satisfied himself that bismuth at least will be retained by the silver.

The most important feature of the results is that some of the ore tested was considered of no value, and if tested by the ordinary crucible method with the usual flux for sulphide ore would give no button. Is it not possible that some good bodies of ore have been overlooked, because the camp assayer reported in all good faith that the ore carried no values?"—J. N. McLeod.—*The Mining World*, May, 1907, p. 622. (H. A. W.)

THE ESTIMATION OF FREE ACID IN THE PRESENCE OF METALLIC SALTS.—"The author in stating that the end point of titrations of free acid in the presence of metallic salts when caustic alkali is used is frequently obscured by the tendency of the hydrates to precipitate, suggests the use of bicarbonate of soda, his method being dependent on the following conditions:—

1. The bicarbonates of these metals are slightly but sufficiently soluble for the purpose of keeping the solution clear, and are neutral to methyl-orange, whereas bicarbonate of soda is strongly alkaline to the same indicator. 2. The red colour of an acid solution of methyl-orange and the blue colour of a sulphate of copper solution are complementary, and when mixed in the proper proportions give an almost colourless solution. For example, if to 100 c.c. of a 3 per cent. solution of pure copper sulphate four drops of a methyl-orange solution be added, a pure yellowish green colour is produced. If to this green solution a $N/2$ solution of sulphuric acid be carefully added, it will be noticed that after the addition of two drops or 0·1 c.c. the solution becomes practically colourless. At this stage the further addition of one drop of $N/2$ acid produces a distinct pink coloration, whilst the addition of one drop of $N/2$ bicarbonate of soda produces a distinct green coloration, the original green of the neutral solution only being obtained after the addition of two drops or 0·1 c.c. of the $N/2$ bicarbonate. The further addition of bicarbonate produces no marked change in colour, and no precipitate until a considerable excess has been added.

The colourless stage of the solution and the remarkably sharp change to pink or green according to whether an excess of acid or alkali is added, gives an excellent indicator in the titration of the free acid in solutions of sulphate of copper, even when containing large proportions of the sulphates of nickel, iron, zinc, etc.

The titration is carried out in the following manner:—Three grammes of the sulphate of copper crystals or such a quantity of the solution to be tested as would contain about that weight of sulphate of copper, is put into a 10 oz. conical flask, and diluted with 100 c.c. of distilled water, and four drops of methyl-orange solution added. If the colour produced is pink, the solution is titrated with $N/2$ bicarbonate of soda until the colourless stage is reached; the amount of free acid in the material tested is equivalent to the number of c.c. of $N/2$ carbonate used, plus 0·1 c.c. If, however, the colour produced on the addition of the methyl-orange is green, then the acidity of the solution cannot be greater than the equivalent of 0·1 c.c. $N/2$ bicarbonate, and in this case the solution is titrated to the colourless stage with $N/2$ acid; the amount of free acid in the solution is therefore equivalent to 0·1 c.c. minus the number of c.c. of $N/2$ acid used.

The determination of the free acid in solutions of sulphate of copper by this method is accurate to 0·04 per cent. of free acid on the weight of sulphate employed, but involves a constant correction of 0·08 per cent. or 0·00245 gm. H_2SO_4 above the actual amount shown by titration with $N/2$ bicarbonate to the colourless stage.

The end reaction is still sharp in 5 per cent. solutions, and if preferred $N/10$ solutions of acid and bicarbonate may be employed, but for all practical purposes the above strengths will be found most suitable for works or laboratory use.

The method is also applicable to salts of nickel or iron, etc.; such, for example, as galvaniser's 'pickle,' in which case stronger solutions may be employed than with coloured salts, such as sulphate of copper or nickel."—EDGAR SIMS.—*Chemical News*, May 31, 1907, p. 253. (A. W.)

ON THE DIRECT QUANTITATIVE SEPARATION OF CHLORIDES AND BROMIDES.—All methods recommended for the direct separation of Cl from Br depend on oxidation applied to solutions of bromides and chlorides, but the oxidants used hitherto have possessed a higher oxidising potential than that of an aqueous solution of Cl, and hence the success of these processes have depended on interrupting the process exactly at the point when all the bromine is liberated. Bugarsky first pointed out the necessity of using oxidants whose oxidising potential should be intermediate between that of chlorine and bromine, and the one selected was HIO_3. This method consists in adding to the Cl' and Br' solution, sulphuric acid and potassium di-iodate, and then distilling

$$KH(IO_3)_2 + 10KBr + 11H_2SO_4 = 11KHSO_4 + 5Br_2 + I_2 + 6H_2O.$$

The Br_2 and I_2 pass off with the steam, and the solution contains the HCl and excess of iodic acid.

This method was modified by the author by increasing the amount of iodate taken, replacing the H_2SO_4 by HNO_3, and removing the excess of iodic acid by a single reagent which would introduce no sulphate or other injurious compound, and the excess of which would require no removal. This method in use for six months gave uniformly satisfactorily results, and also with the determination of Cl in crude bromine.—L. W. ANDREWS.—*Journal of the American Chemical Society*, vol. xxix., 3, March, 1907, p. 275. (J. A. W.)

BICHROMATE TITRATION FOR IRON.* — "L. Brandt states (*Zeit. anal. Chem.*, 1906, xlv., 95-99) that in the determination of iron by titration with bichromate solution, diphenylcarbazide ($CO(NH.NH.Ph)_2$) has the advantage over other indicators of being used in the titrating vessel, and not outside it. The method is as follows: To about 1·5 litres of water in a capacious dish are added 60—80 c.c. of dilute hydrochloric acid (sp. gr. 1·12) and 100 c.c. of a solution containing 10 gm. of manganese sulphate, 55 c.c. of dilute sulphuric acid, 1 : 3, and 5 c.c. of phosphoric acid of sp. gr. 1·7. The iron solution which should contain 0·2—0·5 gm. of iron, and 5 c.c. of 0·1 per cent. solution of the indicator are further added, and the titration with bichromate performed. There is, at first, a bright red-violet colour which finally changes sharply to the green of chromic chloride. The indicator may also be employed in the determination of chromic acid, an excess of standard ferrous ammonium sulphate solution being added, and the process completed as above described. The solution of the indicator in acetic acid should be prepared fresh every few days."—*Engineering and Mining Journal*, April 6, 1907, p. 667. (J. A. W.)

NEW PROCESS FOR WHITE LEAD MANUFACTURE. —"The production of white lead in such a way as to avoid as far as possible injury to the health of the workers has long been a problem with which inventors have been busy. We learn from the *Western Electrician* of April 13 that Mr. Clinton P. Townsend, of Washington, has perfected an electrolytic process which appears to avoid the evils hitherto encountered in producing electrolytic pigments — namely, the irregularity in composition and quality and deficiency in covering power. He has accomplished this by a circulation of the electrolyte successively past the respective electrodes, and preferably in the direction from the anode past or through the cathode. Any type or form of cell may be employed, but the sections must be so formed as to provide between them and the sides of the cell a series of channels. The method of arranging the anode and the cathode with provision for free circulation is described, and a filter-press is proposed for dealing with the electrolyte, which must be made to move freely in the requisite directions. It is pointed out that not only does the electrolyte circulate from the anode past or through the cathode, but the pigment or insoluble compound formed at the anode detaches itself therefrom and is carried upwardly by the electrolyte past or through the cathode and thence to a region outside the field of electrolysis. In operation a lead anode is employed and an electrolyte consisting of any salt, the acid radical of which forms a soluble compound with lead. To this solution is added a soluble carbonate. It is stated that a mixture of sodium nitrate or acetate and sodium carbonate, in proportions approximating ten to one, constitutes a satisfactory electrolyte. The current density depends in some measure on the speed of circulation of the electrolyte, and it may be increased as the flow becomes more rapid."—*Times Engineering Supplement*, May 8, 1907. (J. A. W.)

SULPHUR IN COKE.—"The appearance of the coke offers no characteristics as to its percentage of sulphur. A short description of a method of analysis, unfortunately little known, but both simple and reliable, may therefore be useful :—

Into a steel, or better, nickel, crucible of about 50 c.c. capacity, 11·5 gm. of pulverised peroxide of sodium and 0·7 gm. of coke finely pulverised, or 16 gm. of Na_2O_2 and 0·7 gm. of coal should be weighed. The materials should be mixed thoroughly, and the crucible placed on a small tripod in a beaker, so that between the bottom of the crucible and that of the glass there is a distance of 1 cm. Then the beaker should be partly filled with distilled water, so that the crucible is about half covered by water. In the lid of the crucible there is a small hole through which a glowing wire is conducted to light the contents. After about three minutes, the crucible should be put aside, and the contents dissolved. Thereupon the crucible, lid and tripod should be rinsed well, and taken from the glass. So much concentrated hydrochloric acid should be added that the solution just reacts acid; this is seen at once when the acid is slowly added, as soon as a light green colour appears, and the fluid is quite clear. Then the solution should be boiled, and some drops of ammonia, just in excess, should be added and mixed with 15 c.c. of chromate of barium (solution 23 gm. of $BaCrO_4$, 80 c.c. of HCl concentrated, 920 c.c. of distilled water). One should make sure if there is ammonia in excess, and boil until no smell of ammonia can be perceived; the solution must then be boiled down to a quantity fit for titrating. Then one should filter it, wash the sediment with water, and add 1 gm. of iodide of potassium to the filtrate. Subsequently one should cool down to 30 deg. C., add 5 c.c. of hydrochloric acid and starch paste as indicator, and titrate with a decinormal solution of thiosulphate of sodium.

The course of the analysis after Na_2SO_4 has been formed from the mixture by combination with Na_2O_2, is the following :—

$$Na_2SO_4 + BaCrO_4 = BaSO_4 + Na_2CrO_4$$
$$2\ Na_2CrO_4 + 16\ HCl = Cr_2Cl_6 + 4\ NaCl + 8\ H_2O + 3\ Cl_2$$
$$3\ Cl_2 + 6\ KI = 6\ KCl + 3\ I_2$$

Each atom of sulphur releasing three atoms of iodine.

Special attention, in this analysis, several of which can be carried out within an hour, should be principally paid to two points:—Firstly, that the solution is filtered thoroughly. The filtrate must be absolutely clear; only finest filters should be used and the washing done carefully. If one is not in a great hurry, one should let the solution stand over a little before filtering, so that the sediment granulates a little. The second important point is the preparation of the solution of chromate of barium. This preparation should be made—as chromate of barium to be had on the market mostly has an excess of chloride of barium or of chromate of potassium—and washed well. Traces of either of these impurities mentioned affects the result of the analysis.

It is not advisable to carry out this analysis with coal, as coal mixed with peroxide of sodium causes an explosion when lighted."—A. THAU.—*Iron and Coal Trades Review*, May 24, 1907, p. 1855. (A. McA. J.)

ON FERROUS AND FERRIC SULPHIDES.—The main conclusions are stated as follows :—

1. Precipitated FeS does not react with $Zn(OH)_2$ in the cold or at boiling temperature and only slowly at 160 deg., the products being colourless ZnS and $Fe(OH)_2$.

2. In general, the statement that ferric salts give a mixture of FeS and S, when precipitated in alkaline solution by an alkaline sulphide is erroneous. The product is ferric sulphide which reacts with $Zn(OH)_2$ rapidly in the cold, instantly on boiling to form ferric hydroxide and ZnS.

3. Alkaline polysulphides precipitate ferrous salts as ferric sulphide.

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* See this *Journal*, vol. vi., p. 376.

4. Ferric sulphide readily hydrolyses into ferric hydroxide and hydrogen sulphide at 100 deg. — H. N. STOKES.—*Journal of the American Chemical Society*, vol. xxix., 3, March, 1907, p. 304. (J. A. W.)

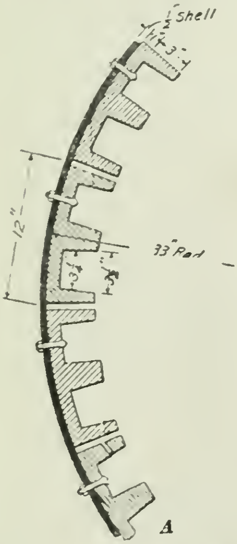
METALLURGY.

SUCCESSFUL TUBE MILL LINING.—"The accompanying illustration depicts what has proved in practical work to be the most successful tube mill lining yet tried. The lining proper consists of a number of hard cast iron segments each 4ft. long by 1 ft. in width with its base properly curved to fit the shell of the given diameter (66 in. in this case).

The base of the shell is of 1 in. thickness while the radial riffles extending longitudinally are 3½ in. apart at their tips, and 3¼ in. at their bases. The segments are laid end to end and bolted to the mill shell of ½ in. boiler iron in the manner shown. When all the segments are in place the usual flint pebbles are fed in and quickly jam so tightly in the riffles that they can be removed in most cases only by breaking out with a hammer.

The illustration shows the interior of a 24 ft. × 5½ ft. tube mill of Allis-Chalmers make fitted with such a lining into which the pebbles have jammed, thus effectively protecting the metal. This mill is one of four in use at the Los Estrellas plant, El Oro, Mex., where the lining was invented. The neighbouring El Oro Co. operates its own foundry where the linings were cast, and this foundry accommodates the other mines of the surrounding region.

The original idea of the inventor was to provide riffles or cogs in the shell which would catch up the pebbles and carry them nearly to the top before allowing them to drop, thus greatly increasing their crushing action on the ore in addition to that of their regular grinding action. Much to his surprise, he found on opening his experimental tube, after several hours of grinding, that the pebbles had wedged tightly into the riffles, thus giving it a lining of pebbles automatic in originally placing itself and in replacing worn pebbles with new ones.



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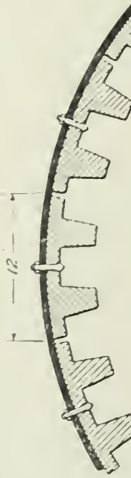

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This lining was thus found to need neither attention nor repairs. Its usefulness and great efficiency having been demonstrated by severe tests, the former silex linings of all four tubes were replaced by the new riffle linings modified from the original design A to the design B now in use. In like manner the other mills of the El Oro region discarded silex linings, and the new lining is now in use at them all. For 8 months, these Los Estrellas linings have been in use in the mills, each grinding 100 tons per day to a pulp, of which about 50 per cent. is finer than 200 mesh and not more than 5 per cent. is 100 mesh.

A recent examination of the original lining at the time of taking the photograph disclosed that the wear on the metal riffles was perceptible only on the rounded outer edges. This slight wear is taken to indicate that the probable life of the metal segments will be from 2 to 3 years, a record yet to be broken." —*Mines and Minerals*, June, 1907, p. 520. (A. McA. J.)

CYANIDE CONSUMPTION DECREASE.—"In regard to saving cyanide in the treatment of gold and silver ores, we have been enabled by a simple method to reduce our cyanide consumption from 4·5 to 1·5 lb. per ton of ore treated, and the results are so interesting that I herewith tender them for publication.

When we first commenced treatment in October, 1905, our protective alkalinity in the working solutions was carried at 0·04 per cent. (in terms of caustic soda), and the cyanide consumption varied from 4·3 to 4·5 lb. per ton of ore.

The ore is valuable for its silver, which occurs as a sulphide, and carries from 12 to 20 oz. of this metal per ton, together with a small quantity of gold. We find that 5·2 lb. of lime per ton is sufficient to keep the alkalinity at 0·04 per cent. Our extraction has been good, averaging 93 per cent. of both gold and silver, and, accepting the fact that a certain amount of cyanide will of necessity be constantly tied up as a double cyanide of zinc and sodium (using sodium cyanide), we have not considered the amount abnormal, though the advisability had been discussed of installing electrolytic precipitation to obviate this loss of potential free cyanide. However, in reading over 'The Cyaniding of Gold and Silver Ores,' by Julian and Smart, I noted what was said in regard to the decomposition of the double cyanide (Na_2ZnCy_4) by an increase in the free alkali. W. R. Feldtmann is there quoted as saying: 'The addition of alkali to working cyanide solutions which have become somewhat weak in alkali, brings up the strength by regenerating, *i.e.*, decomposing, the zinc cyanide so that, as a matter of fact, when the solutions are pretty strongly alkaline they contain no zinc as cyanide, but only the hydrate dissolved in alkali.'

With the object of ascertaining what results could be obtained by increasing the alkalinity, I commenced adding lime, and kept gradually increasing the same until the solutions tested 0·2 per cent. alkalinity. Our working solution for slime, which we carry at 0·125 per cent. KCy, at once began to gain in strength, and kept growing gradually stronger until it showed 0·3 per cent. KCy; the alkalinity was then allowed to fall to 0·09 per cent., when cyanide strength also fell. After many experiments with various strengths, I found 0·135 per cent. to be the least alkalinity with which I could regenerate, and consequently I have kept the alkali at that strength ever since.

As a result of this regeneration, our cyanide consumption has not exceeded 1·5 lb. per ton treated for more than five months; as a matter of fact, no cyanide was added to the slime-treatment solutions for nearly 13 weeks, and the amount used to bring the leaching-plant solutions up to strength was small; this, of course, was due to the large excess of zinc cyanide then existing in the system.

In regard to the testing for alkali, we first determine the free cyanide by titrating with standard silver nitrate solution to the first faint opalescence, without the use of the potassium iodide indicator; we then add a few drops more of the silver nitrate to make the opalescence well defined, and titrate with $N/5$ oxalic acid solution using phenolphthaleine as indicator. This is the method recommended by J. E. Clennell, and as it has been found to check very closely with L. M. Green's ferrocyanide method, and also with that of Gerald Williams,* it has been adopted by preference on account of its simplicity.

Finally, I would say that up to date there is no indication of increase in our cyanide consumption, and as this work has been going on for the past five months, I have come to the conclusion that the regeneration has assumed constant proportion."—L. McN. B. Bullock. *Mining and Scientific Press*, June 8, 1907, p. 719. (A. McA. J.)

Treatment of Matte from the Cyanide Mill.—"It may be of interest to some to know of a quick and complete way in which to extract the metals from a rich matte as obtained at a cyanide clean-up. It often happens that the yield from this source adds considerably to the month's bullion.

Sulphur, the matte-forming element, comes from zinc sulphate ($ZnSO_4$) remaining with the precipitate after acid treatment. The amount of $ZnSO_4$ present will depend, of course, upon the number of water-washes used in the final acid treatment. The more the precipitate is washed, the less will be the matte formed on a gold button. I believe that in our case (treatment of concentrate) it is possible at times for extremely fine pyrites to escape from the filters, and be carried in suspension to the zinc boxes where it is bound to remain with the other precipitate. The amount, however, is exceedingly small. If such be the case after treatment with acid, ferrous sulphate ($FeSO_4$) will be contained in solution, while free sulphur separates out with the precipitate according to the reaction:

$$FeS_2 + H_2SO_4 = FeSO_4 + H_2S + S$$

There is a possibility then, unless a complete roast is given to the acid-treated precipitate, of sulphur being present as a matte-forming material upon melting the final precipitate. The amount of sulphur obtained from this source to form matte will in all probability be slight. Experiments now point to the fact that the amount of matte obtained on a gold button depends principally upon two things, namely, the completeness of the final water-washes after acid treatment and the extent to which the final roast has been carried (whether merely a drying of the precipitate or a complete roast).

The matte that I am treating contains zinc, lead, iron, gold and silver. When using the zinc-lead couple for precipitation, the presence of lead in the matte, if the gold slime is acid treated and then smelted in a graphite pot, is practically inevitable. Lead in the matte may also be due partly to the dissolving (by the cyanide solution) of lead salts from oxidised concentrate followed by the precipitation of lead on the zinc. It may originate from the lead acetate used in some solutions. There are two sources by which iron may enter the matte, either by iron pyrite collecting in the zinc boxes and finally appearing in the acid treated precipitate as $FeSO_4$, or by some iron scale gathered from the roasting pan with the gold precipitate. Zinc seems to enter the matte in considerable amount, especially when the acid treated slime contains some undissolved zinc or has been only partly roasted, so as to contain zinc sulphate.

I believe that a small amount of matte on a button is desirable, for the slag above will, as a general rule, assay less in silver, especially when the matte contains lead. No nitre is used by us to keep down the matte upon fusion of the slime in crucibles. Its damage to the pots amounts to more than the cost of treatment of the additional matte. Previously to using my present method of reduction, we certainly did not welcome the presence of any considerable amount of matte at a clean-up, since the old method of treating small quantities with nitre or scrap iron was slow and incomplete. It is not safe to allow any amount of this rich matte to accumulate (it assays about $20 per lb.), and for that reason the best plan is to treat it immediately at the end of the clean-up, melting the bullion with the gold buttons into bars.

Potassium cyanide acts as a powerful reducing and desulphurising flux, and for this reason it was used for decomposing the matte. Our particular matte is unusually high grade, assaying as much as $45,000 per ton. It contains zinc, lead and iron as sulphides, besides the precious metals. At the monthly clean-up about 800 lb. precipitate is obtained, producing nearly 3,000 oz. bullion, and from the treatment of this we obtain from 30 to 40 lb. matte. The amount varies from month to month. The method of treatment for the matte is as follows:—

The matte, borax and cyanide are put separately through a small rock-breaker and crushed fine. Borax and cyanide are used as fluxes and are put with the matte into three No. 60 graphite pots in the following proportions and order: There are alternate layers of borax, matte and cyanide throughout, until the pot is full, and finally covered with borax. The crucibles are now put into the furnace and a white heat maintained for two or three hours. As a rule, it takes a little over two hours with a good hot furnace. It can be seen when the action is complete, for the charge will subside and the bubbling cease. The action and burning of the sulphur will be violent at the end of the first hour. Also the slag will become quite thick when the action is complete, being removed with a skimmer; this is necessary before pouring the contents of the crucible into a conical mould. If an excess of cyanide be used, it will be found just above the gold button and can be broken off by a blow with a hammer. The matte will now be entirely decomposed, leaving only a light porous slag and the gold button.

This method shows an extraction of 85 to 94 per cent. of the total value of the original matte, depending upon the richness of the material treated."—A. E. Drucker.—*Mining and Scientific Press*, May 18, 1907, p. 638. (A. McA. J.)

Cleaning of Filter Press Plates.—"At the Kalgoorlie mines, where the 'all roasting and sliming process' is in operation, it is found in the subsequent filter-pressing of the pulp that, owing to the presence of hydrated calcium sulphate in the cyanide solutions, there is a tendency for the filter-press plates to become coated with a very hard and extremely adhesive cement, which in time may completely fill the flutings and choke the channels of the plates. The removal of this incrustation has been found difficult and costly, says a report of the Chamber of Mines, Western Australia. The use of pneumatic tools and of 'sand blast' has been resorted to for the purpose. With badly incrusted plates, the latter is the more effective and less costly method of the two, but, costing as it does approximately 20s. per plate in addition to the cost of removal from, and replacing in position, the expenditure is one that causes serious consideration.

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* See this *Journal*, vol. iv., p. 288.

Mr. A. Wauchope reports that at the Great Boulder Main Reef mine early this year it was decided to take advantage of the fact that dehydration of the hydrated calcium sulphate (gypsum) cement would have the effect of removing its cementive properties for the time being, and would leave it in such a condition that its removal would be comparatively easy. To this end an oven was constructed, wherein the plates—about 25 at a time—were vertically suspended, leaving a space of three or four inches between each plate. The roof of the oven was removable, to facilitate the introduction and withdrawal of the plates, and consisted merely of sheets of iron. No fire grate was used, as the object was to produce a comparatively low and uniform temperature, lest strong heating would buckle or crack the plates. All cocks were removed. After 24 hours' heating the plates were removed from the oven and attacked with suitable tools and brushed with brushes made of stout steel wire rope. It was found that the scale was so softened as to permit of its easy removal, both from the fluting and the channels. In addition, all the old rubber rings, blanket strips, and P. and B. paint were removed at the same time. No buckling or other damage resulted to the plates. The total cost of removing the plates from the presses, conveyance to oven, firing, cleaning and replacing in presses, amounted to 10s. 6d. per plate."—*Australian Mining Journal*, June 19, 1907, p. 568. (A. McA. J.)

A STANDARD FOR THE FINENESS OF GROUND POTTERY MATERIALS.*—An elutriation process is considered by the author of this paper to be the only simple method for estimating the amounts of particles of different sizes in a mass of finely ground material. For this purpose, instead of the Schoene apparatus, the following, used by Krehbiel, is recommended. The apparatus comprises three cylindrical copper cans, the bottom of each terminating in an inverted cone of 60 deg. A stream of water is fed into a thistle funnel, which passes to the bottom of the first can, and the overflow from this passes into a similar thistle funnel, and to the bottom of the second can, and similarly from this to the third can. The diameters of the second and third cans are arranged so that the velocity of the stream of water as it rises in the cans decreases, and, consequently, smaller and smaller particles are carried over. The stream is supplied to the first can from a receptacle, provided with an overflow, to ensure a constant pressure. The following table shows (1) the height, (2) the diameter, and (3) the speed at which the water rises in each can, and (4) the size of grain removed :—

	1.	2.	3.	4.
	cm.	cm.	mm. per sec.	mm.
First can ...	15	5·00	1·50	0·04—0·025
Second can ...	15	7·32	0·70	0·025—0·01
Third can ...	15	14·58	0·18	0·01—0·00

The stream from the third can, carrying the milky washings, is allowed to flow away. 20 gm. of the finely-ground material are agitated with water and introduced into the first can, and the stream of water started. Particles whose diameters are above 0·04 mm. are left in the first can, those above 0·025 mm. in the second, and those above 0·01 mm. in the third, and these fractions are afterwards taken out, dried, and weighed, the fraction whose particles have diameters less than 0·01 mm. being estimated by difference. After obtaining the fractions the grains of which have diameters within the limits just mentioned, the fineness of the sample can be expressed as a numerical value, by the 'surface factor' suggested by Jackson, and modified by Purdy.

[The 'surface factor' is obtained in the following way :—Invert the mean diameter of each group. Multiply the 'surface factor' of each group, thus obtained, by the fractional amount of the quantity of the sample having particles of these mean diameters. The sum of these products is the total 'surface factor' for the sample.]

The following examples may be quoted to show the order of the figures of 'surface factors' of some American ground materials :—

Felspar,	Eureka Co.	body spar	1028
	,,	lawned 160	1047
	,,	calcined	1198
Flint	,,	French flint	1031
	,,	,, lawned 160...	1111
	,,	,, re-ground ...	1324

—C. F. BINNS, Transactions American Ceramic Society, 1906, p. 244.—*Journal of the Society of Chemical Industry*, April 15, 1907, p. 321. (W. A. C.)

PANS *v.* TUBE MILLS.—"Some tests carried out at the Dolores Mill, in Chihuahua, Mexico, shew the relative merits of Pans *versus* Tube Mills as sliming machines.

In Test No. 1, the ore was crushed with cyanide solution by 15 stamps, weighing 900 lb., through a 2 mesh screen, the pulp passing to the two Bryan mills with 60 mesh screens, then to four pans in series, thence through two settlers, and finally through two cone-classifiers. The overflow from the classifiers went to the agitation plant, and the underflow was returned to the tube mill for re-grinding, the re-ground product joining the stream from the settlers at the head of the cones.

In Test No. 2, the pulp from the Bryans passed directly to the tube mill, thence to the cones, the overflow as before to the agitation plant, the underflow being returned to the head of the pan series, thence through settlers to cones as before.

In both tests samples were taken every hour, over periods of twelve hours, from the heads of the tube mill and heads of pans, and also from the discharge of same. The bulk 12-hour sample was determined from the proportion of sand and slime in each sample, the difference in slime content between heading and tailing being taken as the sliming power of each machine.

The following are the results obtained, from the average of five samples :—

TEST NO. 1.

	Percentage of slime.		Percentage of slime.
Tube mill heading	8·20	Pan heading ..	11·02
,, ,, tailing	42·64	,, tailing ...	52·54
Slimed by tube mill	34·44	By pans.. ..	41·52

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* See this *Journal*, vol. vii., Sept., 1906, p. 72.

TEST NO. 2.

	Percentage of slime.		Percentage of slime.
Tube mill heading	4·66	Pan heading	7·36
,, ,, tailing	44·08	,, tailing	51·88
Slimed by tube mill	39·42	By pans	44·52

Taking the average of both tests, to bring the machines under the same conditions, gives the following result :—

Percentage slimed by tube mill	36·93
,, ,, ,, pans	43·23
,, in favour of pans	6·09

The pans used are of the Wheeler type, with plain flat muller and dies; they have a diameter of 5 ft. The speed is 65 revolutions per minute. The power required to drive the four pans is 50 h.p. The tube mill is a machine specially constructed by Allis-Chalmers, made in sections for mule transport, and I believe is one of the first, if not the first sectional tube mill made. It has a length inside of 16½ ft. by 3 ft. 6 in. diameter, and runs at a speed of 35 revolutions. It is lined with white iron, and carries a load of 4½ tons of flint pebbles. The power required to drive it is 20 h.p.

Although the above tests showed a small margin in favour of the pans for sliming, yet when the questions of power and cost are considered the tube mill is undoubtedly the more economical machine. The wear and tear in the case of the pans is considerably more than that of the tube mill, and the difference in power required for driving, by itself would more than minimise any advantage the pans have in sliming capacity." ROBERT CLARKE.—*Mining and Scientific Press*, April 6, 1907, p. 431. (T. L. C.)

MINING.

THE DEVELOPMENT OF THE HAMMER DRILL.—"No doubt, those who are interested in mining or the improvements in mining machinery are aware that during the past three years many changes are noticeable in the methods used in breaking ground. Since the introduction of the first piston drills, many years ago, improvements have, of course, been made from time to time, but the general construction has been the same. While admitting that the cost per foot broken was, in many cases, higher than with hand drilling, operators have continued using the larger 3 in. to 3¼ in. machines in their cross-cuts and shafts, and 2½ in. to 2¾ in. machines in their drifts, raises and stopes. At the same time, there have been many complaints made against the operating and maintenance expenses.

Many changes have been inaugurated in the past three years in the methods of rock breaking, and from the subject of this paper you will naturally infer that I intend to show what results have been obtained by the use of the air hammer drill.

It has been proved conclusively that with the improved type of hammer there is very little shock or vibration to inconvenience the operator, and that the remedy does not lie entirely in the improved machines. The conclusion is that, with a gad made with the proper taper, at each blow of the piston the tool is forced into the crevice and remains there until the next blow is struck. This leaves the operator with but the natural vibration of the hammer to contend with. In the later types of machines the piston is made to cushion on live air on the back stroke, and no excessive jar occurs.

A difficulty is developed in the construction of the drill bits. These were made of ⅞ in. hollow staybolt iron, with 6 in. of hollow steel welded to either end for a shank and bit. It was found that crystallisation would take place at the welds and either the shank or bit would jump off in a very few minutes. We turned our attention at first to the steel, and after careful experiments found that the bits with six cutting edges cut faster and held their gauge better than any other style. The cross bit was tried, but was discarded on account of its tendency to 'rifle' a hole and 'hang up' in crossing slips at an acute angle. We also found that with this shape of bit it was hard work to rotate the drill by hand in soft ground, and, while it cleared better, the six-tooth bits gave much better results. Sharpening tools were made for the six-tooth bits, and little difficulty was experienced in drill sharpening after the blacksmith had acquired the knack of spreading the gauge.

Hollow steel was necessary in drilling down holes or flat holes in sticky ground, as it was essential to force part of the exhaust air through the steel to clear the hole of cuttings. The welded bits would not stand in hard rock, and we were in a quandary as to how we would overcome this difficulty, until it was discovered that a ⅜ in. hole could be drilled through a 4 ft. piece of steel in a gun-barrel lathe. We used this class of steel for some little time, until we found that we could purchase a brand of hollow rolled steel in Sheffield, England.

With these changes we were able to drill 4 ft. holes in any ground at a speed varying with the formation of from 1 to 10 in. per minute. In testing the machine thoroughly, however, we found that a cast-steel cylinder would crystallise easily, and we were forced to change the material used in its construction to the best grade of steel shafting, forming the complete cylinder out of the solid metal.

We then found that it was next to impossible to get men who would hold the machine over their heads and drill uppers. One could not blame the drill runners, as it was certainly an awful task to set a man at, and only the very strongest could stand it for any length of time. The screw-feed machine mounted on a light drift column was tried, but finally discarded, when one of the manufacturers constructed an air-feed attachment. With this machine it was possible to 'set up' and 'tear down' in less than a minute, and the drilling capacity of the machine was increased 100 per cent. Machines of this type were placed in all of the larger mines of the country; and it was soon found in competitive tests that they would out-drill any of the 2¾ in. machines.

In the large cross-cut or railway tunnel, the large piston drill still holds the advantage, as it is possible to drill deep rounds and break the ground far more economically and rapidly than would be possible with the small hammer type. Even in this class of work, however, there is a place for the small drill, as they can be used most successfully in block holing and in taking up bottom or trimming up the walls where projections are left, either on account of missed holes, or because the drill runner has miscalculated the load he had on a certain hole. They can also be used for drilling uppers in the roof for pipe hangers, or for putting in short relief holes in the heading, where they will give the cut holes a better chance to break.

In shaft or winze work, however, they can be used for drilling the full round of holes, and even in the larger sized shafts, records show a saving which is simply astounding. To those who are familiar with this class of work, the reason is obvious. With piston drills, even in the hands of the best of runners, more time is lost in setting up, tearing down, lining up, cranking in and out of holes and in changing steel, than is consumed in drilling. Where the ground is even and breaks well, it is not so bad, but where slips are encountered frequently, and time is consumed in preventing a 'hang up,' or where the formation is such that a great many holes must be drilled to break the ground, the air-hammer drill has all the best of the argument.

The machine, steel and hose, can all be put into one bucket, and the drill runner can be at work in five minutes from the time he goes on shift. The steel being loose in the chuck, he loses no time in changes, and he can start a new hole while a piston drill runner would be cranking out. Holes can be drilled in any direction and pointed to take advantage of any slip. No holes need be lost, as it is practically impossible to get a 'fitchered' hole even in the worst kind of ground. All of the holes can be drilled to any depth desired, up to 4 ft., except in very heavy sulphide ore, and the sump holes can be put in any part of the shaft.

When timbering commences, the small hammer drill is indispensable for cutting hitches and taking out projections where they interfere with the sets, and for these purposes alone, will pay for itself in 30 days.

In a drift, a drill runner can either mount the air-feed drill on a column or, where it is possible to pull the cut out of the top, he can work with the air feed alone.

There is a diversity of opinion as to whether it is advisable to use a 2¼ in. piston drill or an air-hammer drill in this class of work, and a decision, from an economical standpoint, can only be obtained by actual tests. As a usual thing, however, the hammer drill can be used advantageously in all drifts where a 4 ft. round, bottom 1½ in. will break without leaving any 'guns.' With a piston drill of the size mentioned or larger, it is possible to make more rapid progress, but the cost per foot will probably be greater.

In ground which breaks short, however, the piston drill usually has the advantage, as a large hole is drilled, and the powder can be got to the bottom of the holes where it belongs.

In stope work the 2¼ in. machine does not have a chance against the air-hammer tool, and the records on some of the Cripple Creek properties show that their stoping costs have been reduced one-half since the introduction of the small machines. Where the values lie in small streaks of very high grade ore, the saving is enormous, as the stopes can be carried even narrower than by hand, as the drill can be run in any place, a runner can get his body.

Not only can more work be done with hammer drills than with piston machines, but the saving in repairs is well worth considering. In nearly all of the small hammer drills, there is but the one moving part, and when properly made, this should last for months. The rotation being accomplished by hand, there are no pawls or springs to break, and no rifle nuts to wear out. As the machines are fed by air, no feed screws or nuts are required. Taken as a whole, with a properly made air-hammer drill, the repairs are less than 25 per cent. of those of a 2¼ in. machine.

Working at a drill pressure of 100 lb., a 2¼ in. machine requires 100 cub. ft. of free air per minute, while an air-hammer drill of the valveless type uses but 25 cub. ft. at the same pressure.

Another advantage possessed by the hammer drill lies in the fact that it is not necessary to employ a first-class drill runner, as a good miner, who knows how to point his holes, will do just as much work after two or three days' experience, as the best man one could hire."—H. L. Sinclair.—*Compressed Air*, May, 1907, p. 4,450. (H. A. W.)

Special Methods of Shaft-Sinking.—"Under the head of special methods of shaft sinking, I propose to consider the methods that have to be resorted to, when ordinary methods cannot be applied economically on account of an excessive influx of water. Such complications are mostly confined to coal mining, and arise but rarely in metalliferous mining, mineral deposits usually lying in the older, less heavily watered rocks.

Year by year, as the coal seams available within the area of the exposed coalfields are more extensively drawn upon, it becomes increasingly necessary to sink down to the so-called concealed coal measures, overlain by more recent formations, many of which carry vast quantities of water. It is a question what quantity of water may be regarded as necessarily involving the use of special methods. The largest amount that I know to have been sunk through successfully was at Horden Colliery, where 9,250 gallons of water per minute at a depth of 540 ft. were dealt with. But in many cases far less than this amount has proved an insuperable obstacle to sinking in an ordinary way. It is possible that in the future ordinary methods of sinking may be found applicable in cases in which they have heretofore been considered impracticable, by the use of suspended tubbings, as introduced by Messrs. Haniel & Lueg, of Düsseldorf, a system which has never been tried in this country, although it has been used with much success in several deep sinkings on the Continent.

The special methods to be used will differ, according as the water-bearing strata to be traversed are firm or running ground. In the former case the Kind-Chaudron method is mostly resorted to. This system was first employed about the year 1854, since when about eighty shafts have been sunk by it, of which five have been in this country. A pair of shafts were put down by it in 1876 at the Cannock and Huntington Colliery in Staffordshire, but the operation failed through the breaking away of the ground at the moss-box in each shaft. Sinkings by this method were also commenced, almost simultaneously, in 1877 at the Whitburn Collieries, and were completely successful. Finally, the shaft of the Consolidated Kent Collieries Corporation, Limited, was sunk by this method at Dover in 1904—5. A depth of 1,095 ft. had been reached by ordinary methods of sinking, and there still remained some 90 ft. of Oolitic and Liassic rocks to sink through in order to reach the coal measures. This was done by the Kind-Chaudron method, and a column of tubbing 1,120 ft. high was put in. The modern continental practice in such cases is not to put in a column of tubbing the full depth of the shaft, but to lower a length of tubbing corresponding only with the depth bored out, and fitted with a false cover as well as a false bottom, the economy being obvious.

Mr. Riemer, one of the leading German authorities on sinking, considers that it will not be possible to execute sinkings by the ordinary Kind-Chaudron method at depths exceeding 2,000 ft., because it would be impracticable to handle cylinders thick enough to withstand the ppressure corresonding to that depth, the limit of thickness according to him being about 5 in. Various proposals for modifying the method, so as to render it available at these great depths, have been put forward by Mr. Riemer and Mr. Thomson, but have not yet been put into practice.

When the ground to be sunk through is running as well as water-bearing, the Kind-Chaudron method is no longer applicable. In such cases three groups of methods are recognised—namely, (1) driving down annular sheet piling, (2) forcing down continuous cylinders of brickwork or iron, and (3) the Poetsch freezing process. Annular piling is the oldest method, and is only applicable when the stratum of soft ground to be passed through is not very thick, and is comparatively near the surface. As the piling is usually in lengths of 15 ft. to 20 ft., and as each new ring of piles has to be driven inside the one above, it follows that the original diameter must be taken very much greater than the finished diameter of the shaft. The method is not often seen now, but a good modern example may be quoted—namely, the recent sinking (September, 1906) at Bowburn Colliery Co., Durham.

It may be here suggested that ferro-concrete sheet piling might be used. Piles of this material, 60 ft. in length, have been driven, and it has also been found practicable to add fresh lengths to the top of piles already driven, and thus to get down to almost any desired depth. Furthermore, the adjacent sides of ferro-concrete piles can be grooved, and when the piles are down, cement can be run into the spaces thus left, making the piling practically watertight. As far as I know, this system has never been tried for shaft-sinking, but there appears to be no reason why it should not be perfectly successful.

The method of sinking continuous cylinders, and excavating the ground from inside them, has only been used in this country for moderate depths. There have been several good examples of shallow sinkings by it in Scotland, where sand, mud, and silt on the seashore have been sunk through for depths of about 100 ft., the sinkings at Musselburgh (1901–2), and at Bridgeness (1878), being cases in point. In a more recent sinking of a pair of shafts at Ardeer (1905), about 80 ft. of silt had to be traversed, and a similar process was adopted, using, however, the somewhat antiquated device of the air-lock (first used by Triger in 1839) for the actual excavation, the men working in compressed air at the bottom of the cylinder. Modern practice seems generally to favour the use of some form of dredge for excavating the material from the inside of the cylinder.

This principle has reached its greatest development in Germany, where it has been extensively used for sinking down to firm ground through the overlying thick masses of superficial deposits. The modern practice consists in sinking a cylinder, usually of masonry, built upon a massive iron cutting shoe, until skin friction prevents its further descent. Tie bolts, attached to the cutting shoe, are built into the brickwork, and by means of these a heavy iron anchor ring is secured to the upper part of the shaft. A cylinder of iron is then built up inside the first cylinder, and is forced down by powerful hydraulic presses, which work against the anchor ring. The ground inside the cylinder is excavated by some form of dredge; grab dredges, bucket dredges, bag dredges, and the air lift all having been made use of.

The freezing method was devised by F. H. Poetsch in the year 1883. Although originally intended for sinking through wet quicksands, it has also been applied to sinking in solid but broken strata, carrying much water. Although it had been used repeatedly with success in both France and Germany, the first applications of this method in Britain have been comparatively recent, and have all been confined to the Durham coalfield. The first example was the sinking of a pair of shafts at Washington in 1902. Freezing was subsequently employed in two very difficult sinkings at Easington and Dawdon. At the former place the operation was a failure; the yellow sands here are exceptionally thick—namely, over 100 ft.; they lie at 165 ft. beneath the surface; the water is brackish, and the water level is affected by tidal movements. At the neighbouring colliery of Dawdon, however, the operations, commenced about the same time, were perfectly successful.

Another very interesting sinking by this method is now in progress at the famous Wearmouth Collieries, originally sunk in 1826. It is now proposed to sink a new shaft close to the site of the old ones. If this were done by the ordinary method, the necessary pumping would relieve the tubbing in the old shafts of the hydrostatic pressure under which it now stands, and when this pressure comes on again, which it would do as soon as the new shaft were tubbed off, the old tubbing would in all probability be destroyed. Hence it has been decided to use the freezing method, which does away with the need of pumping, and, therefore, does not affect the hydrostatic level of the district.

Finally, reference may be made to a method that has only been used once or twice in the north of France—namely, the forcing of cement slurry through boreholes into soft, fissured strata, and thus forming a wall of concrete within which sinking can be performed; this method was used successfully a few years ago at the Lens Collieries."—Henry Louis.—*London Mining Journal* (from Institution of Civil Engineers), July 6, 1907, p. 30. (A. McA. J.)

Breathing Appliances for Mines.—Dr. Boycott begins by referring briefly to some physiological considerations which bear on the efficiency of breathing appliances. The common object of such apparatus is to enable the wearer to remain for a varying time in air which either contains insufficient oxygen to support life (*e.g.*, black-damp), or is poisonous (*e.g.*, after-damp) or irrespirable (*e.g.*, irritating vapours). This is attained by connecting the mouth to a closed space into which oxygen is delivered and from which the carbonic acid is removed. The apparatus must be air-tight as far as the passage of air from outside to inside is concerned. In most circumstances met with in mines, a very small leak does not necessarily render the apparatus useless. The outside air must not be breathed, either because it does not contain enough oxygen (black damp), or because it is directly poisonous (after damp). In the former case a small leak is obviously immaterial: in the latter the occasional leakage of a few cubic centimetres of average after-damp (containing 1 or 2 per cent. of carbon monoxide) into an air space of 4 or 5 litres would cause no material symptoms for many hours. If, however, irritating vapours, such as thick smoke, are present even the smallest leak must be avoided.

The amount of oxygen required varies enormously with the degree of bodily activity, and to a less extent with the individual concerned. A man at rest uses about 0·3 litres per minute, while severe exertion raises the consumption to 2 litres per minute; a degree of activity corresponding to walking 3 miles per hour requires rather more than 1 litre per minute.

The volume of air breathed cannot be indefinitely increased, and the breathing of air which is deficient in oxygen is an imperfect stimulus to deeper or more frequent respiration. It is, therefore, necessary that the percentage of oxygen in the air breathed should be maintained as well as the total quantity available for use. It is undesirable that the percentage of oxygen in the air breathed should fall much below that in outside air (21 per cent.), and it is definitely dangerous to breathe a mixture containing as little as 10 per cent. oxygen, especially while doing muscular work. There is no considerable physiological advantage in increasing the percentage of oxygen in the mixture breathed beyond 20 per cent.

The symptoms resulting from breathing air deficient in oxygen are insidious and indefinite, and vary very much with different individuals. The first obvious symptom, however, especially in persons not particularly directing their attention to the matter, may be sudden loss of consciousness. It is, therefore, absolutely necessary that some definite warning of the failure of oxygen, not dependent on personal vigilance or sensations of giddiness or the like, and preferably of an emphatic mechanical kind, should be given to the wearer of a rescue apparatus before the percentage of oxygen has fallen to a dangerous limit. This is best obtained by arranging and working the apparatus so that it becomes filled with a mixture containing 60 or 70 per cent. of oxygen. Before the oxygen percentage has been reduced to the limit of safety, the apparatus will then become so empty and respiration so impeded (or impossible) that the attention of the wearer cannot fail to be attracted mechanically. Any apparatus which permits the possibility of the user being unknowingly led into breathing a mixture containing less than 12 per cent. of oxygen must be looked upon as extremely dangerous.

In all cases a moderately distensible bag is necessary to act as a reservoir and as a buffer or cushion for the ebb and flow of inspiration and expiration. Such a bag must be so large that the deepest inspiration or expiration may be contained in it quite easily. Expiration or inspiration against even the slightest pressure (as into a full or out of an empty bag) is extremely distressing, and not free from danger if persisted with.

The physiological effects of breathing air containing an excess of carbonic acid are usually obvious. Air containing 5 per cent causes definite panting, which at 6 or 7 per cent. becomes distress; even 4 per cent. produces unpleasant increase in respiration (hyperpnœa), and greatly reduces the capacity of doing hard work. It may be stated in a general way that 3 per cent. is the maximum permissible, while it is desirable that the percentage should not rise much above 1 per cent. Definite symptoms of poisoning do not appear—at any rate in the course of a few minutes—until the percentage rises to 10 or 15.

At the end of each expiration the mouth and larger air passages are filled with the last and most impure air from the lungs; the air contained in this "dead space" between the lips and the lungs forms the first part of the succeeding inspiration. Any breathing apparatus attached to the mouth increases this "dead space" by the volume of air between the lips and the purifiers, or the inlet of pure air. In this way a large dead space may render the air actually inspired very impure Thus with a dead space of 200 cub. cm. containing 6 per cent. carbonic acid, an average inspiration of 600 cub cm. from air containing 1 per cent. carbonic acid contains, as a whole, 2·7 per cent. carbonic acid, while with a dead space of 400 cub. cm., the figure is raised to 4·3 per cent. Generally speaking, therefore, the dead space in the respiration apparatus should not exceed 200 to 250 cub. cm. at most.—Dr. BOYCOTT, Report of Royal Commission on Safety in Mines.—*Iron and Coal Trades Review*, July 5, 1907, p. 31. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

HYDROMETALLURGY OF SILVER. By OTTOKAR HOFMANN. 336 pages, price $4. (Hill Publishing Co.)

"While barrel and pan amalgamation have been steadily practised for the recovery of silver, and the lixiviation of silver ores as well, little has been heard of them outside of Mexico and one or two South American countries. Dating from Stetefeldt's hyposulphite lixiviation, very little if anything new has been added to the literature of the hydrometallurgy of silver, which makes this book on the subject decidedly welcome. The subject matter in this book is separated into two parts: the first dealing with the preparation of the ore, or chloridising roasting, and the second taking up the extraction of the silver from ore prepared for lixiviation or amalgamation. The synopsis and the method followed by the author in dealing with the subject, furnish an index of what can be expected in a book, and then the thought arises: will there be anything new or valuable in the work? A few brief remarks will show that the author is experienced, original, and learned in his subject, thus being qualified to teach others and add something new to the subject. At once, the reader finds that sulphide minerals are divided into three classes, according to the readiness with which they will combine with sodium chloride. This is both a new and valuable systematisation of ores. It has been the prevailing belief that certain kinds of coarsely crushed minerals permit of quicker percolation than finely crushed minerals. This the author, with certain qualifications, states is erroneous, and that increased filtration due to coarse crushing is so slight as to be of no practical value. He then shows that finely pulverised ore is in a much better condition for chemical reactions induced by chloridising roasting and lixiviation, and further that chloridised ore permits quicker filtration than raw ore. Professor Christy, some years ago, made experiments in chloridising roasting, in which the loss from volatilisation was found greatest if salt was added early to the roast. Mr. Hofmann says the proper time to add salt is during crushing operations, provided that the nature of the ore will permit, thus mixing it thoroughly. Reasons are given why lead and zinc sulphides require more salt for a chloridising roast than iron and copper sulphides. The author is not only practised in the subject on which he writes, but has made a special study of chloridising roasting, the results of which he has incorporated in his book. In addition to the valuable experience he relates, he gives practical suggestions that add materially to the value of the work from the standpoint of the man engaged or about to be engaged in carrying out the process."—*Mines and Mining*, June, 1907, p. 504. (A. McA.J.)

THE METALLURGY OF THE COMMON METALS. By LEONARD S. AUSTIN. First edition. £1 1s. (San Francisco, U.S.A.: *Mining and Scientific Press.*)

"This excellent work is divided into ten sections, and deals with the general principles underlying the treatment of ores of gold, silver, iron, copper, lead, and zinc.

Although each, naturally, can receive only a limited amount of attention, the information is clearly put and admirably practical, and the work will be found particularly useful as giving information not obtainable in many more pretentious treatises. Perhaps one may take some exception to the section relating to the occurrence and treatment of gold ores, as it is scarcely so practically treated as the others, but the sections relating to copper and lead are full of useful details, and will prove valuable both to the beginner and to the fully qualified metallurgist.

In addition to the sections dealing with the metals individually, we have special and probably more generally useful sections dealing with generalities such as sampling, fuels, combustion, thermo-chemistry and refractory materials, roasting—a most useful section—and refining. Finally, a section is devoted to the commercial aspects of metallurgy, including notes and advice as to labour and general costs, and a useful schedule of prices ruling at various mining centres for the purchase of ores, concentrates and mattes.

Unfortunately, the work has one inexplicable and absolutely unpardonable fault. It possesses no index."—*London Mining Journal*, July 6, 1907, p. 13. (A. McA. J.)

PRACTICAL COAL MINING. Vol. II. By Leading Experts in Mining and Engineering, under the editorship of W. S. BOULTON, B.Sc., F.G.S., etc. Six volumes. First edition. Vol. II. 6s. (London: Gresham Publishing Company.)

"Volume II. contains the following sections:—

(4) Shaft Sinking (continued).
(5) Breaking Ground.
(6) Methods of Working and Timbering.

The description of the Ordinary Methods of Sinking are continued, and amongst them excavating by hand is briefly described, the absence of which we had noticed in the earlier part of the section. We should have liked a little more practical detail, but this should, perhaps, be looked for in the pit rather than in the text book.

A separate chapter is devoted to Special Methods of Sinking, and fully illustrated. The special conditions which led to the adoption of these methods might have been more fully explained.

Breaking Ground is an eminently practical chapter on explosives, shot firing, and drilling machines: but where is the miner's pick and all the other tools whose uses are generally given under this heading? Perhaps they will come later on.

A short and useful chapter is devoted to Driving Stone Drifts; the subject is well worth it. It is an almost constant occurrence in the life of a colliery, and a very different thing from driving a coal road. Much money is often wasted through not realising this difference, and expecting the ordinary coal hewer to undertake it.

The last chapter in this section deals with Coal-cutting Machines. The main characteristics of the many types of coal cutters now on the market are well and briefly described with illustrations, and the author's general remarks are excellent.

Section 6 begins with a description of the various Methods of Working, of which there are really only two, with endless modifications adopted to suit special circumstances. The general remarks are very much to the point, but no amount of book reading, without experience underground, can enable anyone to select the method best suited to a particular mine. The student has here, however, a number of actual instances, with the sections and conditions of the mine, and the special method adopted, which cannot fail to be of use to him.

Workings under the sea, beneath running sands, bogs, etc., have received special attention, as also boring against old workings; both are attended with special dangers, and the author's remarks will be read with interest.

Timbering needs experience and much practice. The possibilities of saving expense without in any way increasing the dangers from falls, etc., are greater than appears at first sight. We realise the difficulties of explaining these details—small in themselves, but which mean so much—within reasonable limits (viz., notching trees and bars so as to avoid splitting under weight, the angle at which timber should be set to carry side or top weight; in heaving ground, inclined seams, etc.); still we think the subject worthy of more notice in a work entitled 'Practical' Coal Mining.'"—*London Mining Journal*, July 13, 1907, p. 66. (A. McA. J.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 298/07. A. H. Gibson. Improvements in coal mining machinery. 12.7.07.

(P.) 299/07. D. Brazzi. Improved screen for use in gold bearing ore crushing stamp batteries. 15.7.07.

(P.) 300/07. J. E. Smith. A new and improved locomotive smoke box. 16.7.07.

(C.) 301/07. F. H. Maxwell. Improvements in stamper batteries. 16.7.07.

(P.) 302/07. R. V. Driscoll. Improvements in safety devices for mine cages and like vehicles. 19.7.07.

(C.) 303/07. H. D. P. Huizer. Manufacture of a drinking vessel made of ice and apparatus therefor. 19.7.07.

(P.) 304/07. S. S. Lawry. Improvements in apparatus for concentrating loose material containing precious stones or metals. 20.7.07.

(P.) 305/07. A. R. Wojciechowski. An improved process and means for disintegrating and rendering friable diamond bearing ground. 23.7.07.

(P.) 306/07. S. Fraser. Stamp battery mortar box ore feeder. 24.7.07.

(P.) 307/07. J. T. Nicholson. Apparatus for taking finger prints. 24.7.07.

(P.) 308/07. A. Gemmell. Automatic dry earth closet. 25.7.07.

(P.) 309/07. F. Slade (1), C. R. Pratt-Smith (2). Automatically regulated gravitation fire escape. 25.7.07.

(P.) 310/07. A. E. Pack. Improvements in rope jockeys. 25.7.07.

(C.) 311/07. J. Gill. Improvements in rotary motive power engines applicable also to rotary pumps, air or gas blowers, compressors or exhausters, meters and the like. 26.7.07.

(C.) 312/07. C. H. Fox. Improvements relating to railway wagons and like vehicles. 26.7.07.

(P.) 313/07. J. W. Sutcliffe (1), E. W. Nortin (2). An improved grip for attaching trucks to a cable used in mechanical haulage. 26.7.07.

(P.) 314/07. P. T. Morrisby. Improvements in pulp distributors. 27.7.07.

(C.) 315/07. B. Hall. Process for the extraction of gold and silver from ores. 27.7.07.

(P.) 316/07. S. Fraser. Improvements in linings for stamp mortar boxes. 27.7.07.

(P.) 317/07. J. F. Cook. Improvements in means for collecting crushed ore products. 27.7.07.

(P.) 318/07. J. Conning. An improved joint applicable for use between soil pipes and closet pans or stop sinks and the like. 27.7.07.

(P.) 319/07. P. A. Chiappero. Improvements appertaining to internal combustion motors or engines. 27.7.07.

(C.) 320/07. R. Howe-Gould. Improvements in fluid meters. 27.7.07.

(C.) 321/07. A. H. Warmsley. Improvements in apparatus for producing combustible gas from petrol or other spirit or oil. 29.7.07.

(P.) 322/07. W. R. Jones. An improved form of brake with automatic powers to prevent the cage from dropping when the rope breaks. 29.7.07.

(P.) 323/07. A. Harvey. Improved and portable shower bath. 30.7.07.

(C.) 324/07. H. Undeutsch. Improvements in devices for stopping cages, vehicles and the like in case of rope rupture. 2.8.07.

(C.) 325/07. C. H. Alston. Improvements in or connected with internal combustion engines. 2.8.07.

(C.) 326/07. E. Surcouf. Improvements in motor road vehicles and trams. 2.8.07.

(P.) 327/07. A. M. Bell. Ironing machine. 3.8.07.

(P.) 328/07. J. Craig. Apparatus for treating diamond bearing deposits, ores and the like with liquids. 6.8.07.

(C.) 329/07. H. W. Blaisdell (1), H. A. Brooks (2). Improvements in or relating to filter leaves. 7.8.07.

(C.) 330/07. J. R. Brown. Improvements in or relating to lining for grinding mills. 7.8.07.

(C.) 332/07. H. Poehl. Improvements in machines for washing coal and other materials. 9.8.07.

(C.) 333/07. H. Poehl. Improved method of and apparatus for draining washed coal and other materials. 9.8.08.

(P.) 335/07. J. McIntosh. Improvements in chucks of rock drilling machines and the like. 10.8.07.

(P.) 336/07. J. McIntosh. Improvements in the valves of rock drilling machines or other reciprocating engines. 10.8.07.

(P.) 337/07. S. S. Osborn. Improvements in centrifugal amalgamating apparatus. 10.8.07.

(P.) 338/07. H. Pakeman. Machine for developing motive power with cold water and compressed air. 10.8.07.

(P.) 339/07. J. E. Wheatley. Movable clamp bar bracket. 12.8.07.

(P.) 340/07. J. E. Wheatley. An improved apparatus used to filter compressed air after exhausting or escaping from rock drill machines or other devices. 12.8.07.

(P.) 341/07. J. E. Wheatley. Apparatus for purification of air after blasting. 12.8.07.

(P.) 342/07. J. E. Wheatley. Suction filter pan. 12.8.07.

(P.) 343/07. J. E. Wheatley. Apparatus for purifying air. 12.8.07.

(P.) 344/07. R. H. Paterson. Improvements in and relating to steam traps. 13.8.07.

(C.) 345/07. E. L. Ransome. Improvements in apparatus for progressively moulding plastic material in conduits *in situ*. 13.8.07.

(C.) 346/07. J. E. Friend. An improved rotary gas engine. 15.8.07.

(P.) 347/07. W. Boulton. Improvements in the fixing of pulleys, cams and the like to shafts. 15.8.07.

(P.) 348/07. C. A. Jeffreys. Improvements in pipe couplings. 15.8.07.

(C.) 349/07. H. Lee. An improved rotary engine. 15.8.07.

(C.) 350/07. F. C. Brown. Improvements in apparatus for the treatment of ores and for the electrolytic deposition of gold and silver and other metals from solutions containing such metals. 16.8.07.

(C.) 351/07. F. Russell. An improved field gate. 16.8.07.

(C.) 352/07. C. M. Chamberlain. Improvements in filters or extraction apparatus. 16.8.07.

(P.) 353/07. H. T. Durant. Improvements in and relating to the separation of solids from liquids. 16.8.07.

(C.) 354/07. F. L. Lane (1), The Leeds Forge Co., Ltd. (2). Improvements in wagon doors. 16.8.07.

(P.) 355/07. F. Simon. Improvements in rope clutches for mechanical haulages. 17.8.07.

(C.) 356/07. H. I. Seemann. Improvements in amalgamators. 17.8.07.

(P.) 357/07. H. B. Johnson. An improved jockey for endless wire haulage. 19.8.07.

(P.) 358/07. J. Wilshir (1), G. Muller (2). Improvements in hose couplings. 20.8.07.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

BYERLEY, B. G., *l/o* Johannesburg; Bulawayo Club, Bulawayo, Rhodesia.

CARTER, J. W., *l/o* Germiston; Crown Deep, Ltd., P. O. Box 102, Fordsburg.

EVANS, G. S., *l/o* Gwanda; *c/o* Mr. J. Harris, Alma Cottage, West Horsley, near Leatherhead, Surrey, England.

MACKAY, A. N., *l/o* Boksburg; Caley, Wimbledon Park, London, S.W.

MCEWAN, T., *l/o* Belingwe: Matabele Sheba G. M. Co., Ltd., Essexvale, Rhodesia.

TOOMBS, C., *l/o* Johannesburg; P. O. Box 80, East Rand.

WILLIAMSON, REG., *l/o* Fimiston; North End Mines, Ltd.. Kalgoorlie, W.A.

THE JOURNAL

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OF SOUTH AFRICA.

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VOL. VIII. SEPTEMBER, 1907. No. 3.

Proceedings
AT
Ordinary General Meeting,
September 21, 1907.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, September 21st, Prof. J. Yates (President), in the chair. There were also present :—

40 Members: Dr. J. Moir, Messrs. F. Alexander, R. G. Bevington, W. R. Dowling, K. L. Graham, A. Heymann, J. Littlejohn, G. O. Smart, Prof. G. H. Stanley, A. Whitby, H A. White, W. Beaver, H. D. Bell, Dr. J. T. Carrick, E. H. Croghan, Andrew F. Crosse, N. M. Galbraith, J. Gray, J. C. Greer, Jr., H. R. Jolly, W. H. Jollyman, J. A. Jones, A. Kressmann, G. A. Lawson, Hy. Lea, J. Lea, C. W. Lee, G. Melvill, J. E. Metcalf, J. T. Milligan, D. J. Pepler, J. F. Pyles, O. D. Ross, H. A. Scarf, S. Shlom, J. A. Taylor, J. P. Ward, H. Warren and H. Wiley.

19 Associates: Messrs. S. J. Cameron, J. W. Carter, J. Chilton, W. M. Coulter, C. E. Deakin, C. L. Dewar, B. G. Gerardy, J. H. Harris, O. Harrison, J. W. Hawthorne, C. B. Hilliard, G. F. Jones, R. W. Leng, R. K. Maxwell, C. G. J. Moore, C. B. Simpson, W. Waters, A. Waugh, and L. J. Wilmoth.

11 Visitors and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as published in the *Journal*, were confirmed.

NEW MEMBERS.

Messrs. Gray and Jollyman were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been duly elected, as follows :—

CUNNACK, JAMES T., Knights Deep, Ltd., P. O. Box 143, Germiston. Assayer.

GILMORE, ARTHUR DAVID, Robinson Deep G. M. Co., Ltd., P. O. Box 2584, Johannesburg. Cyanider.

LINDSAY, ROBERT, Ferreira Deep, Ltd., P. O. Box 1056, Johannesburg. Cyanider.

SYDENHAM, CHARLES THEODORE, Knights Deep, Ltd., P. O. Box 143, Germiston. Assayer.

WALLACE, ALEXANDER, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Engineer.

The Secretary announced that the following gentlemen had been admitted Associates by the Council since the last general meeting.

AVENT, ALEC., Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanider.

BURT, WILFRED, Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Sampler.

COGLE, CHARLES THOMAS, c/o W. J. Wolhuter, Esq., 37, Mint Road, Fordsburg. Mining Engineer.

MURRAY, GEORGE DALY, Ivanhoe Gold Co., Western Australia. Metallurgist.

ROBERTSON, JOHN RANKINE HASTIE, Rezende, Ltd., and Penhalonga Proprietary Mines, Ltd., Penhalonga, Umtali, Rhodesia. Cyanide Manager and Assayer.

THORBURN, FREDERICK HARRISON, Nigel G. M. Co., Ltd., P. O. Box 20, Nigel. Assayer.

WILLIAMS, GERARD W., Ida H. G. M. Co., Ltd., Laverton, West Australia. Analytical Chemist.

GENERAL BUSINESS.

The Secretary read a letter from Mr. Chris. Toombs disclaiming all intention of having made personal reflections on any member of the late staff of the New Goch G. M., especially on Mr. C. L. Johnson, the former cyanide manager, now in the employ of the same mining group on another of their mines.

Dr. J. Moir: As your representative on the Seymour Science Library Committee—the Committee which deals with selecting books for that library—I wish to announce that the Committee has lately increased the usefulness of the library by allowing the books to be taken out for home reading: but in order to cover themselves against loss, the Committee will require a deposit to the value of the books taken out. There are, of course, a number of the books which cannot be easily replaced, and the Committee therefore reserves the right to refuse to lend such books out; but every other facility in reason to use the books will be freely given.

The President: In view of the fact that technical books are very expensive nowadays, I fancy that many of you will look on this change as a boon: you can get very little in the way of technical literature for two guineas, and I think the idea of the library lending the books out is a very satisfactory innovation. I may say incidentally that the ultimate home of the Seymour library will be at the Transvaal University College, but that will not mean that the public will not have access to it as at present; it will be freely open to the public just as it is now. Might I suggest to you, as members of this Society, that you should propose books to be purchased for the Seymour library? I am sure that Dr. Moir and the Committee will be glad if you will come forward and mention any books which might be added to the collection with advantage.

We have received a very interesting letter from our member, Mr. Stokes, of the *Rand Daily Mail*, who is, as you know, going round the world visiting the various mining fields, and he has offered to read a paper descriptive of his journey which will be illustrated by lantern slides. I need hardly say that the Council has accepted this offer with thanks, and as it is to be an illustrated lecture, we hope to have a special meeting, held at some hall suitable for lantern exhibitions.

THE PATENT OFFICE.

Mr. W. H. Jollyman: On two occasions Patent Office legislation has formed the subject of discussion at meetings of this Society. Will you permit me to add a few remarks concerning what this Society has called "this deplorably retrograde" step of our new Government?

This Society has objected to the Ordinance passed a month ago, mainly on two counts; first, that it is essential in order to inspire confidence in the breasts of would-be inventors that their specifications should be examined by a Commissioner possessing wide scientific and legal attainments; secondly, the increased cost to inventors. Would it not, I ask, inspire the necessary confidence if specifications were to be examined by two gentlemen, one of whom would be well trained in law, and the other would possess scientific knowledge? It seems to me that such an arrangement would be better than if both the legal and scientific validity of a specification were to be adjudicated upon by a scientific lawyer. Of such a man we should not be surprised to hear that lawyers considered him a good scientist, but not up to much as a lawyer, while scientists would be likely to disown him at the same time acknowledging his legal ability. You may be interested to learn that the Comptroller of Patents in England is neither scientist nor lawyer, whilst in America the Commissionership is a political plum. The late Commissioner in this country was a most estimable gentleman, an excellent lawyer, a Master of Arts and an eminent ornithologist. Under the new arrangement, Mr. Juta—Advocate of the Supreme Court, and Barrister of the Inner Temple—will attend to the legal part of the business, while Mr. H. D. Wedlake—formerly Analyst to the Explosives Office, who has had practical experience in mining matters in England, and has for many years been associated with technical chemistry, and is an inventor himself—will adjudicate upon the scientific soundness of patent specifications.

Seeing that the list of subjects for which patents are applied for contains not less than 146 classes, it is clear that the technical adviser will have his work cut out, but if instead of the dual arrangement a scientific lawyer were at the head of affairs, we might get a lot of law, but probably precious little science.

Regarding opposed cases, which will be tried by a Judge of the Supreme Court (not by the Supreme Court itself, as erroneously suggested by our late President), I do not think we need fear injustice resulting from this arrangement.

Regarding cost to inventors, one speaker with a powerful imagination, foresees costs increased ten or even a hundred times. What a pity he did not read Mr. Juta's addendum to the Report of the Patent Office Commission! It is to the effect that the cost of litigation in the Court of the Commissioner of Patents was as great as, if not greater than, in the Supreme Court. It is possible, however, that the costs will in future be somewhat greater in opposed cases, unless Patent Agents are permitted by law to instruct counsel direct instead of through the intermediary of attorneys. So far as I am aware this point has not yet been settled.

Now, regarding the work of the Commissioner of Patents, it is interesting to note that he had the control of three branches:—

1. The grant of Letters Patent.
2. The Registration of Trade Marks.
3. The Registration of Companies.

The percentage revenue of his office from patents work varied from 4 to 11 during the years 1902—1906. The number of applications in any one year since the war has not been greater than 550, and about three-quarters of the applications are usually granted. The number of oppositions dealt with between 1900 and June, 1906, was 54. I could supply you with more figures, but these are sufficient to show that the patent work of this country is small in amount, though varied in character, and does not demand the services of a highly-salaried Commissioner.

The most regrettable feature of the whole affair is the effect that this Society's action in making an ill-considered and ill-advised protest must inevitably have upon the position the Society assumes in the opinion of the Government. Not so long ago, we were informed that the incorporation of the Society was greatly to be desired, one of the reasons advanced being that our representatives could then approach the Government with the claim that we had a definite position in the eyes of the law as a scientific body. Our first action after attaining our legal majority is to impress upon the Government our inability to see both sides of a simple matter; our arguments have been found wanting; we have been snubbed by members of our Legislature, and we have deserved what has been meted out to us.

There is one other aspect of the case to which I would like to direct your attention. I have already told you that the number of oppositions is not great, but since inventors have to consider very carefully the cost of fighting an opposed case, this Society would be doing a more useful service to inventors if, instead of sending those "Help-us-or-we-perish" telegrams to the Hon. Members of the Legislative Council, it were to use its influence with the Chamber of Mines—the inventor's greatest bugbear — to secure for inventors more consideration at the hands of those, whose power and wealth enable them to crush the life out of an invention.

The President: Do you desire to make any motion on this matter, Mr. Jollyman?

Mr. W. H. Jollyman: The only thing that I should like to suggest is that since Patent Office legislation is coming up again next year, the Council would do well to appoint a sub-Committee to ascertain all the facts before approaching the Government again.

Mr. Andrew F. Crosse: I should like to know if the Governments of this country and of England look upon this question of patents merely as a means of squeezing money out inventors, or whether they look upon it as a means of encouraging industries. I am afraid that in England as well as here, patents form a convenient way of increasing the revenue. There are many honest men here who invent things, but who have no possible means of finding if their inventions are original or not. One may think he is working on something new when it is not new. We want the Government to say to us in effect: "My good fellow, this thing has been worked out already." Instead of that they take our money and do not help us in any way. If I pay so many pounds the Government should give me reasonable protection. In Germany and America they do take a certain amount of trouble, but in England they don't. I do not think the Government has any right to take money from a man, when they know perfectly well they are simply robbing him of his money because his invention is not a patent at all. I do not say that any Government should guarantee a man against an action at law, but they should afford him the information if a substantially similar idea has already been patented.

Mr. R. G. Bevington: I should like to amplify Mr. Crosse's remarks with regard to preventing would-be inventors from wasting their money. Some 13 years ago I patented a certain invention here and I wished to patent it in America. The application was forwarded to America, and I received back from the Patents Office there a very courteous letter enclosing me cuttings and diagrams from various sources, showing me that my would-be invention had been anticipated there. This was done at absolutely no cost to myself. Not only did the American Patent Office send me a very courteous letter, but they went to the trouble of sending me illustrations showing the device, which absolutely covered what I wished to patent, without any cost to myself. I would like to see a department here which would keep in touch with every invention, if possible.

The President: I am sure you have listened to Mr. Jollyman's remarks and have found them interesting, but personally I really must say that I take exception to his stating that our protest was ill-advised. Of course, he is welcome to take any view he likes of the matter, but certainly the protest was backed by the unanimous voice of your Council, and by the South African Association of Engineers, and by the Mechanical Engineers Society, and therefore it is practically unanimous so far as the scientific societies of Johannesburg are concerned, and Mr. Jollyman stands alone in his views. His suggestion regarding the appointment of a sub-Committee of Council has already been carried out, for such a Committee already exists.

Prof. Stanley showed me to-day an interesting little matter, and I proposed that he should come along this evening and place it before you also.

NEW TEST FOR CASSITERITE.

Prof. G. H. Stanley: I wish to bring before the members a new test for tinstone, discovered recently by Mr. Weber, Curator of the Geological Museum, Johannesburg, which I think will be of interest.

The test is as follows:—It consists simply in placing the piece or particles of material to be tested in contact with metallic zinc and covering with fairly strong hydrochloric acid, when cassiterite becomes coated in a minute or so with a thin layer of metallic tin, which, if the particle is of sufficient size to be so treated, takes a polish on rubbing between the fingers.

Powdered material should be examined afterwards under the microscope, when an otherwise apparently barren sample may be seen to contain specks seemingly altered to metallic tin.

For pieces or powder, a small block of zinc into which a shallow hole half an inch or so in diameter has been drilled is convenient, and the same test may be applied to polished faces of hand specimens or microscope sections by using zinc dust or powder spread on the surface and then applying acid.

Specimens prepared as above were exhibited, one showing very clearly intergrowths of tourmaline and cassiterite in a polished hand specimen, and at the close of the meeting a demonstration of the mode of application was given.

Mr. W. H. Jollyman: What is the strength of the hydrochloric acid used?

Prof. G. H. Stanley: It does not matter much; say, half concentrated acid.

Mr. A. Whitby: In connection with these experiments of Prof. Stanley's, I wish to remark that the test proposed may find a use in field work. The insufficiency of the methods of detecting tin in some minerals, such as titanic iron and tourmaline are well known to prospectors, and it seems to me highly probable that the test might easily be made an extremely useful one where doubt exists as to the actual presence of tin, and particularly when it is associated with titaniferous iron. Ordinary prospecting methods of concentration frequently fail to show tin in this mineral when it has been conclusively proved to be present by analysis.

Mr. R. G. Bevington: Prof. Stanley mentioned this matter to me some few weeks ago. In order to try the method I placed a portion of a crushed sample then in my possession, which was supposed to contain tin, upon a sheet of zinc, and covered it with hydrochloric acid. To the eye it showed no difference, and I had doubts whether there was any tin present. I then took the sheet of zinc and placed it under a microscope, and on examining it I found several crystals most beautifully coated with tin and fully bearing out Prof. Stanley's statement.

VISIT TO JUMPERS MINE.

The President: I wish to call the attention of the members to the proposed visit to the Jumpers mine next Saturday, concerning which I have been asked to say a few words on this so-called resueing. You remember that Mr. Richardson touched upon it in his reply to the discussion on his paper on mine subsidence. It is a method of working which is particularly applicable to thin reefs, more especially where there is a well defined bedding plane under or over the reef allowing the waste rock to break away easily and cleanly. Supposing you had a mine with a 6 in. leader, it is obviously impossible for you to take out that leader without stoping some of the ground which lies alongside it, and generally speaking, the smallest stope you can work in is something like 3 ft.; I do not wish to say that you cannot work in less than 3 ft., but generally speaking, 3 ft. is looked upon as the minimum practicable size. Let us assume that you have a reef merely 6 in. in thickness and give it a value of 12 dwt. If you carry this leader in a stope of 3 ft. it follows that you are mining 2 ft. 6 in. of waste. You are diluting the ore to 1 of reef with 5 of waste. Allowing for sorting one-half of the waste you have 15 in. of waste and 6 in. of reef going to the mill, or 21 in. milling width, and you get a value in your mill of 3·4 dwt. as the result of your mining a 6 in. leader having an original value of 12 dwt. 3·4 dwt. is not a payable proposition and that is where resueing comes in. In the system of resueing you do not dilute the ore; you first take out the waste above or below it, and then you get your reef with its value practically undiluted. Resueing gives you for your mill the 6 in. of 12 dwt. value more or less.

Mr. H. A. White: Have they got boys of a special size on the Jumpers?

The President: The angle of the reef is about 40°, and, therefore, in a 3 ft. stope there is no difficulty in getting about. At Mansfeld, in Germany, the excavations are in many places only 20 in. high, and the men do their work lying on their side and holding rock drills of the pneumatic hammer type, whilst the stuff is taken away from the working place by boys who crawl along on "all fours." I merely mention this as an instance showing what can be done in the way of stoping small, if it is necessary.

EXPERIMENTS IN FIRE ASSAYING AT THE REDJANG LEBONG MINE, SUMATRA.

By G. B. Hogenraad (Associate).

The nature of the samples sent to the assay office, may be judged from the following analyses, made by Mr. J. E. Clennell, June, 1905.

		Battery Pulp.	Sands Charge.	Slimes.
Silica	SiO_2	83·33	86·02	76·17
Alumina ...	Al_2O_3	7·07	5·31	9·09
Ferric oxide ...	Fe_2O_3	3·24	2·93	3·54
Calcium oxide ...	CaO	·92	·65	1·54
Sulphur ...	S	·62	·60	·59
Magnesia ...	MgO	·73	·29	1·38
Manganese oxide	MnO	·13	·08	·31
Copper oxide ...	CuO	·04	·01	·05
Zinc oxide ...	ZnO	·02	·03	·07
Selenium ...	Se	·02	·03	·02
Moisture at 100° C.	H_2O	68	·44	2·00
Undetermined ...		3·20	3·61	5·24
		100·00	100·00	100·00

Up till November 29th, 1906, the fluxes adopted for the assays were as follows :—

For one A.T. charges (assayed with four A.T. flux).

	%	Weight per Charge. gm.
Carbonate of soda ...	59·26	69·14
Fused borax ...	19·75	23·04
Litharge... ...	19·75	23·04
Charcoal... ...	1·24	1·45
	100·00	116·67 = 4 A.T.

For two A.T. charges (assayed with 190 gm. flux).

	%	Weight per Charge. gm.
Carbonate of soda ...	67·18	127·65
Fused borax ...	16·54	31·42
Litharge... ...	15·50	29·45
Charcoal ...	·78	1·48
	100·00	190·00

In both cases the lead buttons produced weighed 20 to 22 gm.; they were cupelled at once without scorification. The slag was black and absolutely opaque, moreover so deliquescent, that on standing a few hours it became nearly fluid. The results were fairly regular for the gold, but for the silver the difference was sometimes as much as 1 mgm., equivalent to a difference of 10 dwt. silver for the two A.T. or 20 dwt. for the one A.T. charges.

Mr. Clennell found that by decreasing the amount of borax in the flux, the amount of silver shown by the assays increased. This was one reason for trying a flux without borax. Another reason was that the slag produced with the old flux was, on the points stated above, not satisfactory. So experiments were started with a new flux, based on the following points :—

First, it was thought that the amount of charcoal for each charge was too high : the following equation

$$2PbO + C = 2Pb + CO_2$$

shows that 12 gm. of charcoal yield 413 gm. of lead, so 1 gm. charcoal must yield 34·4 gm. lead. This was verified by several trials, always resulting in 34 to 35 gm. for 1 gm. charcoal.

Second, it was known from experience that equal proportions of soda and litharge gave a good and clean slag.

Third, the use of borax was considered detrimental. After several trials it was found that the best fluxes for our samples were :—

For one A.T. charge (assayed with 100 gm. flux).

	%	Weight per Charge. gm.
Carbonate of soda ...	50·00	50·00
Litharge... ...	49·55	49·55
Charcoal... ...	·45	·45
	100·00	100·00

For two A.T. charges (assayed with 200 gm. flux).

	%	Weight per Charge. gm.
Carbonate of soda ...	50·00	100·00
Litharge... ...	49·85	99·70
Charcoal... ...	·15	·30
	100·00	200·00

The buttons produced weighed about 30 gm.; the slag was of a nice green colour, clear, transparent, and quite dry even after long standing. Some trouble was found at first from lead sticking to the mould. This was due to the slag being so very fluid that the heavy lead got clear and was first in the mould, whereas with the old flux a part of slag ran in first, solidified and prevented the lead from adhering to the iron. With the new flux the trouble was overcome by rubbing the mould inside with powdered lime.

Everything went all right for some time until a new trouble occurred. After heavy rain at night next morning many crucibles cracked in the furnace, even when dried all day before. This was only the case with the two A.T. charges, and seemed to be simply due to the big amount of litharge in the crucible. Upon experiment it was found that the following flux gave no more trouble, and it was therefore adopted.

		Weight per Charge· gm.
Carbonate of soda ...	55·00	110·00
Litharge... ...	44·85	89·70
Charcoal ...	·15	·30
	100·00	200·00

The results with the new flux were very regular, especially for the gold, agreeing to within ·02 mgm., but for the silver the results were not so satisfactory, until it was noticed that with buttons weighing more than 30 gm., necessitating scorification before cupellation, the silver nearly always agreed to within ·1 to ·2 mgm. This suggested scorifying all the buttons before cupellation, and this was adopted. It was found that about 15 minutes' scorification gave the best results.

Beginning with November 29 the new fluxes with scorification were adopted for all the assays with the following results :—

Sample.	Date Nov. 1906.	Old Flux Dwts. Gold.	Old Flux Dwts. Silver.	New Flux Dwts. Gold.	New Flux Dwts. Silver.
Battery pulp	13	24·8	155·2	26·0	160·0
Slimes	14	18·9	166·6	19·5	169·0
Battery pulp	14	22·8	153·2	23·3	160·2
Sands charge	14	22·5	146·0	22·9	150·1
Battery pulp	20	23 6	149·4	24·6	158·9
Sands charge	20	25·2	145·3	25·4	150·6
Slimes	22	27·4	237·6	29·8	247·7
Battery pulp	22	43·3	278·2	42·5	254·5
Sands charge	22	41·2	265·8	43·0	272·0
Average		27·74	188·6	28·55	191·4
In favour of new flux :				+ ·81	+ 2·8

This table points very distinctly to the fact that borax really carries gold and silver into the slag, and the discontinuance of its use with our ores is therefore a necessity.

Though more gold and silver were recovered by assaying with the new fluxes, it sometimes happened that the proportions between gold and silver in the one A.T. charges (battery pulp, sands charge and slimes) were wrong, so that the assay had to be repeated. Till this time no correction for loss of silver by volatilisation in the muffle or absorption in the cupels had been made. As it was now considered that one assay ton was too small a quantity for the assay of our daily samples, we started on December 18 to run all the samples, battery pulp, sands charges, slimes, mine residue, and slag samples in two A.T. charges with the two A.T. flux, and with the first three descriptions of samples a check was run, consisting of gold and silver wrapped in 30 gm. lead foil, to control any loss during scorification and cupellation.

As for the gold there was practically no loss, so that weighing gold for the checks was very soon stopped. Taking 10 to 15 mg. silver the loss was 0·9 to 1·0 mg. in the back row, 0·8 to 0·9 mg. for the next row in the muffle, and so on, equivalent to an increase of 9·0 to 10·0 dwt. silver for the assays in the back row, 8·0 to 9·0 dwt. for next assays, and so on. With these checks so run there are now no discrepancies in the proportions between gold and silver in these samples.

The residue samples seldom give any trouble, but as they are treated through the whole assaying without checks, it is a wise precaution to scorify and cupel them in the front part of the muffle.

In addition to all the advantages the new fluxes offer in assaying, they are from an economical point of view cheaper than the old fluxes, as the saving in borax more than makes up for the cost of the extra litharge used. Compare, for instance, 40 kilos old flux (= 20 kilos of one A.T. flux + 20 kilos of two A.T. flux) with 40 kilos of the new two A.T. flux.

	Old Flux. Weight. Kilos.	Old Flux. Price.	New Flux. Weight. Kilos.	New Flux. Price.
Carbonate of soda ...	25	£27·50	22	£24·20
Litharge	7·2	3·75	17·94	9·40
Borax ...	7·0	6·75	—	—
		£38·00		£33·60

The prices are taken from Baird and Tatlock's catalogue.

To summarise : —

The new flux gives bigger buttons, a clean fluid slag, very satisfactory assays and is cheaper than the old flux.

The President: I am always particularly pleased when we hear from our foreign members, and I move that a very hearty vote of thanks be passed to Mr. Hogenraad for his paper, which comes to us from Sumatra.

DISCUSSION.

Mr. F. G. Guthrie: The analysis of the ore shows the lode is clearly a quartz reef—containing over 80 per cent. SiO_2, small quantities of Al_2O_3 and Fe_2O_3, and traces of other metals, none of which are in sufficient quantity to interfere with the fluxing of the ore. With such a simple ore the difficulty experienced in obtaining a suitable flux by the assayer is a little difficult to understand. The only element to be taken into consideration is the SiO_2, and as 1·5 of soda flux is about equal to 1 of SiO_2, I fail to see wherein lies the difficulty of choosing a flux giving a readily fusible slag and one yielding all the gold, if not all the silver. Speaking from experience, it is only under exceptional conditions that one can hope to obtain anything approaching the true silver contents in an ore, by the pot assay.

It will be seen from the proportions of flux and ore, as originally used by the assayer, that he used no less than four parts of flux to one of ore, proportions which only highly complex ores justify. Further, his soda is double the ore, which can only render the slag pasty, for he only uses sufficient PbO to yield a 22 gm. button, leaving none to act as a flux, whilst his borax (23 gm.) is excessive. The correct proportions for the above ore would be 1 of ore to 3 or 3·5 (at the outside) of flux, made up as follows:—

Ore	1 A.T.
Soda	1·5 A.T.
PbO	1·0 ,,
Borax	·5 ,,

Little or no addition of flour or charcoal need be made, the ore containing sufficient sulphur to reduce a 15 gr. lead button, and leave a proportion of the PbO to act as a flux, amply large enough for one A.T. charge of ore. In the two A.T. charges the soda is again in needless excess. While the PbO is quite insufficient to both yield a lead button of at least 20 gm., and also act as a flux, for it is essential that a fair proportion of PbO should pass into the slag if a good liquid slag is wanted—and if the slag is not very liquid and easily poured the assay is likely to be unreliable. With the excessive soda used, if PbO was not allowed to act as a flux, it would be quite impossible to obtain a liquid slag.

The excessive deliquescence complained of is probably due to a very moist climate, and nothing more astounding. The same is noticed during the wet season here, or in Rhodesia, though not to the same extent.

Though the assayer found his original flux gave fairly regular gold values, he seems surprised that his silver values differed from 10 dwt. to 1 oz., according as one or two A.T. charges were used. But if he would bear in mind for a moment the many ways it is possible to lose Ag during the assay by the "pot" method—loss in slag, by volatilisation, and by absorption by the cupel—he would at once see there was really nothing to be astonished at. Bearing in mind also the many sources of error in the pot assay for Ag, the method is to be unhesitatingly condemned, and however applicable in theory the method may be, in practice it is found sadly wanting, and most misleading. Assays done in triplicate, or duplicate, will give beads of almost the same weight, leading the unwary assayer to imagine the results are therefore correct. In some particular cases, as when assaying an ore containing metallic Ag, the "pot" assay undoubtedly gives the actual value of Ag contents, but in the majority of cases the values obtained by this method are much below the truth, for unless the metal occurs in the metallic state, oxy-sulphides and other complex compounds are formed during fusion, which pass into the slag. Consequently it is only possible to obtain results, at best only approximately correct, by refining the slag one or more times, which in a large assay office, where every moment is of value, is quite impossible.

The only "fire" method for Ag approaching correctness is the scorification assay, and even this method, if the ore be excessively rich in Ag is liable to give incorrect results if the slag be not cleaned. If worked, however, with care and all reasonable precautions taken, very satisfactory results should always be obtained. That the scorification assay does actually yield Ag values very much higher than the pot assay can be very easily proved by "running through" a pot assay of highly pyritic deeper level Rand banket, and duplicating by the scorification method. By the latter process I have occasionally obtained bullion beads yielding even 50 per cent. and more of Ag, while the bullion obtained from the pot assay gave a steady 10 per cent. of Ag. The gold will be found to be as nearly identical as the personal element permits. The same holds good for slags, black sands, etc., the difference being particularly noticeable with pan furnace slags and black sands. With these two products the difference in favour of the scorification assay runs into many ounces. And assayers when assaying samples of bye-products, etc., for the gold and silver values, would be well advised to determine the Ag contents by the scorification method. Should they have previously assayed for gold and Ag by the pot assay, they will be considerably surprised at the difference between the Ag contents yielded by the two methods.

Not being satisfied with his results and flux, the assayer proceeded on the right lines by reducing his proportions from 1 of ore to 4 of flux down to 1 to a little over 3, by reducing his soda from 1 to 2 to 1 to $\frac{3}{5}$, and increasing PbO to the same amount, and entirely omitting the borax, which seems altogether a wrong policy. This policy is one which many well-known Rand assayers seem inclined to adopt, but for what reasons it is impossible to understand.

If the pot assay for Ag, under the best conditions, could be relied on, there might be some grounds for its complete omission, for it is quite feasible to suppose that borax does tend to increase the Ag contents of the slag to a slight extent, but, as under the best conditions, when assaying ores containing Ag other than in the metallic state, Ag will always be carried into the slag, the only benefit accruing from the absence of borax seem to be a slight decrease in the inevitable error when assaying Ag ores by the pot method. But when assayers assert that borax also carries gold into the slag, one cannot help thinking that it is simply a feasible excuse for conveniently accounting for the presence of gold in the slag, which is really due to the use of incorrect fluxes. For though the scorification assay always yields substantial Ag, increases over the pot assay, rarely, if ever, do the gold contents differ to any material extent. And the difference is as often a minus as a plus one, and undoubtedly due to the personal element in working with such small quantities as ·1 or ·2 A.T. As a further negative proof, the slags of a good flux seldom contain more than a minute trace of gold, and, more often than not, yield nothing.

It is also maintained that owing to the low fusing point of borax the charge is liable to fuse at too low a temperature, thereby allowing gold to pass into the slag. If experienced assayers would only do a little experimenting before making these rash statements, a lot of unnecessary trouble would be saved the unfortunate young assayer learning his profession. A fair proportion of borax should be used in every pot assay for gold, the amount depending upon the nature of the ore. For an ore consisting essentially of quartz ·5 A.T. to 1 of ore is ample, but as oxides and sulphides and lime, etc., increase so should the borax up to 1 or even 1·5 parts.

The advantages of the presence of borax are several. It increases the fluidity of the slag, which otherwise could only be obtained by a large increase of PbO. And by absorbing the base metal oxides, it keeps the lead button soft and malleable. Further it reduces the base metal oxides of iron and manganese to a lower state of oxidation in which they seem to yield more gold than when in the higher state of oxidation.

Instead of trying to find some occult reason for his pots cracking after rainy weather, why not connect the cracking of the pots and the rain and take precautions accordingly? Store the pots in a dry place, preferably the furnace room. This simple precaution taken, the assayer would find no further cause for complaint, provided he refrained from placing cold pots into a red hot furnace, and used pots by a reliable maker.

In the table giving the gold values obtained by using the old flux and new flux it will be observed that on ore averaging 27·74 dwt. the difference in favour of the new flux is as much as ·81 dwt. This simply goes to prove a very well-known fact, that while almost any sort of flux will yield most of the gold, it is only by experimenting with various fluxes that an assayer can hope to obtain one yielding all the gold.

Finally, the assayer on the mine would be well advised, and find it much quicker if he stopped the double operation of potting and scorifying each sample, and adopted the scorification assay as his usual method. It would assure his obtaining correct Ag values as well as correct gold values in a shorter time. Of course, in samples of low grade gold values the pot assay is a necessity, for large quantities of ore must then be employed.

Mr. H. A. White: The analysis of ore exhibited is somewhat similar to that of Rand banket, and the differences shown are of no importance from the assayer's point of view. It is obvious that no particular reason for the use of borax is indicated. With the author's conclusions I quite agree in this one respect, but I cannot follow him in some other ways. For example, too much flux was used in his earlier attempts with borax, and he is even now using more than necessary. For 1 A.T. of ore 3 A.T. of flux is enough, and for 2 A.T. only 5½ A.T. are required.

The difference of 5 per cent. less litharge and 5 per cent. more soda does not account for the effects observed on pots and the trouble is to be located elsewhere as Mr. Guthrie points out. If I were in the author's position I would experiment with litharge alone as a flux, and by that means might evade the necessity of scorification altogether. It is usually advisable to secure at least 20 gm. of lead button for each assay ton of ore, and the necessary reducing agent should be added as required and not be mixed up with the bulk of the flux before hand. The use of borax need not affect this question at all, as a button of a suitable size can always be secured by the addition of a reducing agent and sufficient litharge.

Mr. Guthrie has not made out a good case for the use of borax, though his idea of it as a reducing agent is novel and will probably remain unique. He is misled with regard to 50 per cent. silver in ordinary Rand banket gold, and would find by experience that 10 to 12 per cent. is the usual range. His scorification results must therefore be attributed to other causes, and I would suggest that he examine carefully the litharge such results are obtained with.

I will not reiterate the experimental foundations for my aversion to the use of borax under ordinary mine conditions here as they may be found by those interested in my contribution to the discussion on Mr. Whitby's paper on routine assaying. In conclusion, I would like to thank the author for his useful paper and to express a hope that it meets with a full discussion of the main issue.

Prof. G. H. Stanley: I have listened to Mr. Hogenraad's paper with considerable interest, more particularly as it is the first I remember dealing with the assay of gold ores containing selenium. I think the author uses the term "flux" very loosely to cover a mixture containing other things.

The old mixture contained far too much fluxing material, at any rate for one A.T. of ore, and even the present reduced quantities appear to be larger than necessary. With the old charge, also, too much charcoal certainly was used, there having been more than enough to reduce all the lead present, and the weights given show that this actually did occur, *i.e.*, 23·04 gm. PbO should yield 21·3 gm. Pb and buttons of 20 to 22 gm. were obtained. On the other hand, I do not think a 22 gm. button sufficiently heavy for two A.T. ore, and more litharge should therefore have been used.

In consequence of the excess of charcoal used the sulphur and selenium in all probability remained unoxidised, keeping up silver and gold in the slag, whereas with the new charge only about one-third of the lead is reduced, the remainder being left in the slag, and this probably has more to do with the increased yield of precious metals than the absence of borax.

The slag in the latter case is obviously well oxidised—a very necessary and well-known condition for the complete recovery of silver—and with regard to the gold, it is common knowledge that excess of lead oxide in the slag is necessary in assaying telluride ores, and the author here presents evidence that the same condition becomes necessary in assaying ores containing the similar element selenium.

In this connection, I would like to point out that the beneficial effect of the new charge was even greater with the gold than with the silver results, the percentage improvement averaging 2·84 per cent. for the gold and 1·47 per cent. only for the silver. I doubt, however, whether such a large excess of litharge is necessary for thorough oxidation, and think the soda also could be reduced, rendering the slag less corrosive, and increasing the life of the pots, any decrease in fluidity being met by the addition of a small proportion of borax, to the use of which, in moderation, I confess some partiality, in spite of its recent denunciation by Mr. White as an "*effete superstition.*" All this, obviously, would have to be proved experimentally to have no bad effect on results.

On a few points I would ask the author for further information. The equation given by the author shows that 1 gm. pure carbon reduces 34·4 gm. lead, and his experimental trials verify this, but I should be surprised to learn that chemically pure carbon had been used, and should therefore have expected a smaller weight of button. Personally I reckon about 25 gm. lead to be reduced per 1 gm. charcoal, and a well-known text-book states that charcoal can be expected to have about 80 per cent. of the theoretical reducing power of carbon. To what then was the reduction of this large quantity due? Again, it is still more surprising to find that in the latest charges 0·3 gm. charcoal, together with 0·36 gm. sulphur (equivalent to 0·14 gm. charcoal) in the charge—in all 0·44 gm. charcoal—reduced buttons of 30 gm. weight instead of about 15 gm. to be expected from the result of his trials. There must be some explanation; was there any reducing matter in the ore other than sulphur and selenium? Further, it is difficult to see why, from such an oxidised slag, the buttons should need scorifying; what was the harmful constituent in the buttons to necessitate this?

I cannot understand the explanation given of the new slag running more slowly from the pot than the lead, namely, that it was more fluid than the old, which did not give that trouble. My experience has been exactly the opposite, the pastier the slag the more difficult it is to prevent the lead running out first.

In conclusion, I think the results of the author's use of checks important, as showing the necessity for working with checks whenever silver values have to be determined, and, perhaps most important of all, he has, even if unintentionally, demonstrated the absolute necessity of making up assay charges with regard to well-known scientific principles and not entirely empirically.

Mr. Andrew F. Crosse: There is only one question which I would like to mention. My experience has been that most of us have discarded scorification, which I do not believe in, except in special cases like silver.

NOTES ON SOME RECENT IMPROVEMENTS IN TUBE MILL PRACTICE.

(*Read at April Meeting, 1907.*)

By KENNETH L. GRAHAM, M.I.M.M. (Member).

DISCUSSION.

Mr. R. G. Bevington: During the early days of the discussion on Mr. Graham's paper I mentioned that I was running some tube mills with a graduated lining, and I promised that when I had the data I would give the results of the running of these linings. Two tube mills, 22 ft. × 5 ft. 6 in., were lined with silex blocks 8 in. thick, about 9 ft. from the feed end and the lining then graduated down to 6 in. in thickness. One mill ran for 198 days, the other for 161 days. The lining of both mills wore through in about the same place, viz., about 12 to 13 ft. from the feed end and from there to the discharge end, where not actually worn through, the silex was very thin, but towards the feed end of the mills the silex was still some 2½ to 3 in. thick. It was very noticeable on the various occasions upon which the mills were opened up for inspection, that the large pebbles, or rather pieces of banket, were found to be at the discharge end and the small pieces at the feed end, quite contrary to what usually obtains in a mill lined with an even thickness throughout. I can only attribute this to the funnel shape which the graduated lining gave to the interior of the mills causing the larger pieces to roll downhill as it were.

A Member: At one end of the mill the pebbles probably had the right height of drop, and at the other end they slid, which would probably account for the difference in the wear of the lining.

Mr. R. G. Bevington: The large pebbles were found at the discharge end of the mill and the small pebbles at the feed end. That was noticeable in both mills.

The President: I would say the explanation is that the large and heavier pebbles would naturally try to reach the point at which they are at the greatest distance from the centre of the tube mill.

REPLY TO DISCUSSION.

Mr. K. L. Graham: When placing my paper before you, I did so mainly with the object of eliciting an opinion from those who were in a position to speak authoritatively on the relative merits of "banket" and pebbles as grinding agents in our tube mills. In this I was to some extent successful, the result of the discussion being a verdict in favour of reef, both on the grounds of economy and efficiency. Moreover, inasmuch as the subject induced two recruits to enter the ranks of our critics I feel I have accomplished something.

Mr. Bevington is the only one of my critics who produces figures in support of the theory that reef is at least as efficient as pebbles for our purpose. I wish he had been able to supplement these figures with results obtained when using pebbles alone other conditions being the same; these data are most necessary, if definite conclusions are to be arrived at. I feel gratified, however, that his conclusions generally are the same as my own.

I regret to have to take exception to Mr. Alexander's opening remarks. He says, "I am naturally much interested in Mr. Graham's paper, particularly so, as the improvements alluded to were first made by myself on the Crown Deep in 1906, where I invented a mechanical device for feeding a sufficient quantity of reef into the tube mills, etc., etc." Why he should suppose that the fact of his having invented a device for feeding reef to his mills, which, by-the-bye, proved useless for the purpose for which it was intended, justified him in appropriating my paper wholesale in this manner I fail to understand.

That the "mechanical device" he mentions, could be the justification for this predaceous attack, I can hardly believe, for by no stretch of the imagination could it be described as even useful, since it was not automatic in any sense of the word, and certainly saved no labour. In fact, from what I saw at the Crown Deep and elsewhere, it was quicker and much cleaner to throw the lumps of quartz into the tube mill through the hollow trunnion in the old way than to use a device, the capacity of which was limited to one lump per revolution of the mill. Nor does he trouble to produce one single shred of evidence in support of this self-complacent announcement, consequently, as it stands, his statement could hardly look more ridiculous. It is frequently the lot of the inventor, whether his invention be merely a safety pin or the most intricate piece of mechanism, to be subjected to attacks from the unfortunate individual who always "has used it before." Having felt over and over again, that but for an inexorable fate pursuing me, in this respect I must have been many times an Edison, I can, in this case, from the happy position of the "inventor," offer Mr. Alexander my sincerest sympathy, who, no doubt, feels himself similarly deprived.

As far as the *use* of reef is concerned I made no claim to be first in the field. I did claim, however, to be the first to come forward with reliable figures obtained from a fair trial, run under working conditions, over a long period, which proved the equality if not the superiority of reef to pebbles as grinding agents. Turning now to his subsequent remarks on his experimental run, I notice that in comparing the feeds of reef and pebbles to the respective mills Mr. Alexander has misread my figures. The quantities given by me, viz., 171·5 tons and 10 tons were for the month of January and not for the 81 days of the trial. On this basis the feed of reef was 5·7 tons and of pebbles 670 lb. per day. His consumption of 2 tons of Danish pebbles per 24 hours was certainly an astounding performance. Like Mr. Kennedy I should have been interested to hear how long this was kept up, and also whether Mr. Alexander has any theory to advance in explanation of such an abnormally heavy consumption ; at this rate his pebble bill would have run up to £350 per month per tube mill—a strong argument indeed in favour of reef, were such a consumption general. In regard to liners, I still remain unconverted. That greater skill is already in evidence in the cutting of the local chert, I quite admit, but that it will outlast the average imported silex has yet to be proved. The set of 6 in. × 6 in. × 4 in. silex blocks put into our Davidsen mill with diamond cement, mentioned in my notes, has now run 165 days, the cost of the liners landed here being £152, equal to 18s. per day of life, and I doubt very much whether it pays to quarry and cut local chert even at £1. Before leaving Mr. Alexander's criticism, one more point appears to me worthy of attention. He speaks of a rise in temperature of 20° F. on the pulp entering and leaving the tube mill, whereas under normal conditions I have not been able to record a higher difference than 7° F. I should like to have heard from Mr. Alexander whether the reading was taken during the ordinary running of the mill, as I cannot but think that abnormal conditions must be the explanation of this phenomenon.

To Mr. Laschinger I undoubtedly owe an apology. In looking over his remarks in our *Journal* of June, 1906, I was not sufficiently discriminating. His words contained a suggestion only, and under the circumstances I think I may be forgiven for considering this tantamount to a recommendation.

I turn with a great deal of pleasure to Mr. Lea's criticisms, as some of the points he raises are of considerable interest. He appears to think there is a tendency to sacrifice efficiency to quantity of quartz fed in our tube mills. If such be the case it is undoubtedly a move in the wrong direction, as I think, it is generally understood, that this class of mill is neither economical or efficient as a primary crusher. He attributes his being able to keep his loads well over half on 2 or 3 tons of reef a day to the fact that he feeds it larger than is the practice elsewhere. I do not entirely agree with him here, the amount required being rather a function of the quantity of pulp fed to the mill than of any other condition. My contention is borne out by the figures he gives subsequently, namely, 390 tons per tube mill for 24 hours, this being from 25 to 30 per cent. more than I find it desirable to put through our mills so as to obtain the best results. I regret that I was unable to include in my notes figures showing the work done by the secondary grinding plant, as represented by the difference in the gradings of the pulp leaving the stamps and the final pulp to cyanide works. Unfortunately, owing to the variation in the screen used at the time for grading analyses on the different mines, such figures would have been useless for purposes of comparison. I believe standard apertures of ·01 and ·006 in. are now in general use, which will greatly simplify matters in future. I differ from him again, when he proposes to feed reef up to 7 in. or 8 in. in diameter. Even allowing merely for the sake of argument, that his theory as to the lesser amount required is sound, if, as I am convinced from actual trial, as high an efficiency can be maintained by feeding a larger quantity of, say, 3 in. to 5 in. in diameter, why struggle to reduce it? Any extra labour entailed can be compensated for by the installation of suitable mechanical appliances, making it as easy to handle 5 or 6 tons of the smaller reef as it is to feed 2 or 3 tons of large lumps by hand ; the monthly profit will be proportionately augmented by the increased tonnage put through. It appears to me absurd therefore to attempt to reduce it, unless for weightier reasons than those given. Moreover, the effect of introducing lumps 7 in. or 8 in. in diameter into the mills would, I think, be the opposite to what he supposes, as they would reduce the effective life of the smaller lumps. Again, if the point that a certain amount of this work is done by impact is conceded, the desideratum in the direction of weight is that sufficient to crush the particles of sand by gravity, anything over the requirement meaning wasted energy. To sum up, I prefer lumps of 4 in. to 5 in. diameter if it is possible to get them ; if not, I take what I can get with the object of keeping my loads well up over half, which after all is a *sine quâ non* if good work is to be done. With his remarks on the necessary size of aperture in the discharge plate I cordially agree. After several stoppages shortly after starting our plant, caused through choked holes, I had them all bored out to ¾ or ⅞ parallel, and have never had trouble since. As I mentioned in my notes, the discharge of small pebbles may amount to ¼ ton a day per tube mill, and as they must be quite inefficient, I always argued that the sooner they were got rid of the better.

As to the advisability of using diamond cement for relining, I must admit that in the light of what Mr. Lea has done with ordinary cement I would not advocate the practice on the score of economy. For patching with only a few hours to do the work, it is incomparable. That the long life of the silex liner now running in our Davidsen mill is to some extent due to its use, I feel pretty sure, it being very noticeable that the joints wear almost as well as the liner itself. Yet it would have to go another 35 days before we would be compensated for the heavy initial cost. Under certain conditions, pertaining, I confess, rather to the exigencies of management than to the every-day economics of tube milling, I still think the money would be well spent, if, as Mr. Lea points out, the tube mill could be got running again in, say, 30 hours.

Mr. Kennedy, with 15 months' experience of reef, still seems to think that pebbles would be better. He fails to support his contention, however, further than to remark that he cannot see how an irregularly shaped lump can be as effective as a smooth round pebble. Many of us shared similar impressions until actual trial disposed of the theory, any data at present available certainly pointing in the opposite direction. In supposing that No. 1 mill was insufficiently fed at 670 lb. pebbles per 24 hours, I think Mr. Kennedy argues from wrong premises, the weight of pebbles required, as I remarked in replying to Mr. Lea, depending more on the feed of pulp to the mill, and on the physical character of the pebbles themselves than on any other condition. I have retained the load at well over half in a newly lined tube mill of similar dimensions with less than 670 lb. of pebbles per 24 hours. In regard to the amount of work done by impact and grinding respectively, I do not think that the experiment carried out by Mr. Kennedy proves or disproves anything, and consequently brings us no nearer to the settlement of this debatable point. I fail to see how any practical conclusions can be drawn, when the conditions under which the experiment was run, were so entirely different from those actually existing, the behaviour of the load against the smooth inner surface of a glass bottle, being no criterion as to what it would do in one of our silex lined tube mills. I merely advanced the theory as possibly accounting for the higher efficiency shown when reef was employed. I am certain, from actual observations, that a certain amount of work is done by impact, especially with a well-worn lining, the circumferential speed of the outer layer of lumps being then nearest to theoretical requirements; how much would be a very difficult question to settle.

In conclusion, Mr. Kennedy mentions a liner recently tried in Mexico with great success, a description of which appears amongst the abstracts in our *Journal* for August.* I agree with him in thinking that there are great possibilities in front of this liner, as it appears well suited to our local requirements. The principle, I believe, was patented some time ago in this country by one of our well-known metallurgists, but so far as I know, no trial of any practical importance has been given it here. So successful does it appear to have proved in the Los Estrellas mills, that it has displaced all other tube mill liners in the El Oro region. It would be interesting to see it tried here, since such a liner, if successful, would mean a big reduction in our tube milling costs.

A FEW NOTES ON THE REFINING OF BASE BULLION.

(Read at May Meeting, 1907.)

By C. W. Lee and W. O. Brunton (Members).

DISCUSSION.

Mr. Andrew F. Crosse: I should like to make a few remarks on this. We should not try for too much. Supposing we have valuable metals, copper, silver and gold—you must not object to my saying copper because on the Rand you have no copper, but where I am at present we have more copper than gold and silver—I take it that it is a mistake to try and get rid of the copper. When I got to the mine where I am now located, I found a large number of Winchester quarts containing nitric acid. This was used to dissolve the copper out of the base bullion and, of course, with it an enormous amount of silver was also dissolved which accompanied the copper down the creek. It is a pity to refine base bullion in that way. You can ship your copper, silver and gold home. I have had a good deal to do with the refining of metals, and it is a question whether it pays to refine them here if you have to get rid of a large percentage of a metal which is valuable. I say, do not do it, send it to England or Hamburg or somewhere else and get value for it.

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* See this *Journal*, vol. viii., August, 1907, p. 60.

THE UTILISATION OF WASTE HEAT IN SLIMES SETTLEMENT.

(*Read at June Meeting, 1907*).

By A. SALKINSON (Member).

DISCUSSION.

Mr. H. A. White: On p. 379 of the second volume of our Proceedings is a statement by one of our Past-Presidents which is directly at variance with the results achieved by Mr. Salkinson. It reads as follows:—

"With regard to the heat factor, it is known to practical workers on slimes that they settle better in cold weather than in hot and in cold water than in hot." This, however, no doubt, refers to the ultimate result when the time available is practically not limited, but even then my experience is not in accordance and agrees with the author's statement that ultimate results are scarcely affected, and the only practical effect of warming the slimes solutions is to hasten the process of settlement. No doubt, the underlying reason is the reduction in viscosity which amounts to about 2 per cent. per degree centigrade at ordinary temperatures.

I have found that in some cases the addition of lime has a similar effect, and the increase of alkalinity to ·03—·05 per cent. will in certain specific cases be of great advantage where capacity is limited. The peculiarity is that the earlier part of the settlement is much accelerated, and this fact renders the method of use when the conditions are suitable and care is used in the close observation required. I find that no disadvantage arises in the zinc boxes from this use of lime, and solutions up to ·07 per cent. alkalinity show faint traces only of gold, leaving the boxes on 40 A.T. assays. In making a considerable number of experiments with glass tubes, 1 in. in diameter and 7 ft. long, and in other small glass vessels I have been struck with the fact that the results obtained are rarely borne out in practice in large tanks. For example, the settlement of slime of various depths from 1 in. to 7 ft. is practically uniform in 1 in. tubes, though in practice the greater depth yields a much denser product. Observations are therefore better made on working tanks and a float may be adjusted to follow the settling slimes and can be read off at suitable intervals. The settlement curves so arrived at are quite unlike similar results in glass tubes, but are, of course, quite reliable.

It is necessary to have waste heat available in order to take advantage of Mr. Salkinson's useful paper, but the removal of cooling towers from the battery water supply is a simple thing and could readily be done in many cases. At the May G. M. Co. we have used the condenser water from the battery engine for some long time for milling purposes, and though at this time of the year the water is about 30° C., no evil effects are observed and the slimes plant gets a great deal of benefit.

I should be glad if Mr. Salkinson, in his reply, would give us the percentages of dry slime in his settled slimes as well as the depths measured, as that will, I think, form a better basis of comparison by which others may judge the results and estimate how far in this direction they may go when additional heat must be paid for.

With reference to the general question of slimes settlement I think the following variable factors, arranged in the order of their relative importance, all require some careful study in order to account for the eccentric results which are quite characteristic of the process.

1. The chemical composition of the slime.

In this connection I find that what is known in this district as the bastard reef yields more slime and a greater clay content than the other reefs, and the slime settles worse.

2. The mechanical composition.

Where narrow reefs are being worked more waste quartzite is milled, and this has a tendency to reduce the clay content of the slime and to cause better settlements.

With regard to factors 1 and 2 conjointly it may be remarked that the presence of organic matter or such colloids as ferroso-ferric hydrate and possibly colloidal silica are very detrimental to effective settlement.

3. The temperature of the solution.

The temperature effect apparently becomes of increased importance in the neighbourhood of 20° C.: which is illustrated by the greater effect shown by the rise from 19° to 20° than from, say, 10° to 11° C., equal charges being compared, and the average hourly temperature noted.

4. The effect of dilution.

I find that the speed of settlement is very markedly affected when comparisons are made between pulps varying between 3 per cent. and 30 per cent. slimes. On some mines the ratio 3 solution to 1 of slime is used, but the ratio 4 to 1 gives a quicker settlement and a smaller residue of undissolved gold.

5. The alkalinity.

If this is varied by means of addition of lime I find the settlement improved: the alkalinity should vary inversely as the average temperature to obtain the best results where time is very limited. In this connection the result of experiments on narrow vessels or tubes are curiously discordant with tank measurements.

Experimentally the final settlement in, say, 24 hours is always best, where alkalinity is lowest down to ·005 per cent. (reckoned as NaHO). At first the tubes with most lime show quickest settlement, but at 12 to 15 hours they are overtaken and badly beaten in the end. No such results are observable on the large scale, and the apparent effect is to accelerate settlement, generally, though the earlier period still shows the greatest improvement.

6. The temperature gradient.

This has a doubtful effect, as the variation in temperature is greatest in the coldest part of the year. Convection currents are caused when the temperature slope has a critical value, below which the result is not significant, but above which it may become of importance. This will probably affect the question of the economic addition of heat to our solutions.

7. The strength of solution in KCN.

An increase in cyanide strength has apparently a small prejudicial effect on the settlement.

8. The shape of the vats used.

The wider and deeper tanks are better for final settlement densities. Width, of course, promotes speed, and depth apparently increases the density of the settled material. In this connection it would seem that a conical bottom is not so effective as a flat one.

I merely give the above factors as they appear to me to invite further observations; any definiteness of statement where observation shows widely varying effects ignores the fact that anomalous results can only be due to want of knowledge of all the varying elements, and I have not attempted definite theories but suggested subjects for further investigation.

Mr. E. J. Laschinger: I read Mr. Salkinson's paper with interest, as it revived in my mind a problem which had occupied my attention either a year or two ago, during one of our winter seasons, when the difficulties of slimes settlement manifest themselves.

The whole problem may be treated generally by asking:

I. Would it pay to install an independent heating plant to warm the solutions circulating in the ore reduction plant and maintain them at a fixed temperature?

II. To what extent might the waste heat of the various power plants on a mine be effective for the purpose of warming the circulating waters of the milling and cyanide plants?

In answer, I submit the following:—

I. Independent heating plant using purchased coal.

Cost of heating a ton of water 20° F., say, from 60° F. to 80° F. 2,000 lb. heated 20° F. = 40,000 B.T.U.

Average Transvaal coal has a calorific value of, say, 10,000 B.T.U.

Efficiency of a good heating apparatus, say, about 70 per cent.

Effective B.T.U. per lb. coal burnt = 7,000.

Pounds of coal required to heat 1 ton of water $= \frac{40}{7}$.

Delivered cost of coal = 140d. per ton.
Handling and firing charges = 24d. ,,
Ash handling = 6d. ,,

170d. per ton.

Cost of heating 1 ton of water $\frac{40}{7} \times \frac{170}{2000} = 0{\cdot}48$, say, ½d.

Cost for heating water in mill per ton ore milled:
8 tons of water @ 0·5d. = 4d.

Cost of heating solution in sands plant:
Per ton of sands treated.
2½ tons of water @ 0·5d. = 2¼d.

Cost of heating solution, 1st and 2nd transfers:
7 tons of water @ 0·5d. = 3½d.

Assuming 75 per cent. sands and 25 per cent. slimes, the total increase in cost per ton milled would be:—

$4\text{d.} + {\cdot}75 \times 2\frac{1}{4} + {\cdot}25 \times 3\frac{1}{2} = 6\frac{1}{2}\text{d.}$, nearly, say, 6d.

In the slimes treatment alone on 2 dwt. slimes the extraction would have to be bettered by 3½ per cent. to pay for the heating of the solutions of the 1st and 2nd transfers.

In the figures on the cost is not included any interest or redemption charges on the cost of installing the heating plant nor cost for extra pumping.

II. To what extent might waste heat be made effective in heating mill water or cyanide solutions?

In most plants on the Rand the use of steam power is confined to the mill engine, compressors and hoisting engines with here and there a steam pumping plant or electric generator plant. The rest of the power used round the mine for the various smaller power requirements is generally electricity, the waste heat of which is not available for the purpose we are considering.

For each ton of ore milled per hour there would be required, roughly:—

For mill engine, 12 h.p. @ 16 lb. steam per hour	192 lb.
For compressor, 6 h.p. @ 18 lb. steam per hour	108 lb.
For hoisting ore from 2,000 ft. depth, 2 h.p. @ 40 lb. steam per hour ...	80 lb.
For handling of men and materials, ⅔ h.p. @ 45 lb. steam per hour ...	30 lb.
	410 lb.

The waste heat of the steam would be, roughly:

1,000 B.T.U. per lb. or 410,000 B.T.U. per ton of ore milled.

To heat the whole of the circulating waters 1° F. would require :—

In mill 8 tons of water per ton milled
In sands plant, say, 2 „ „ „
In slimes plant 2 „ „ „
—
12 „ „ „

$12 \times 2,000 = 24,000$ B.T.U., therefore the solution could be heated $\frac{410,000}{24,000} = 17^{\circ}$ nearly.

All the above calculations are, of course, only approximate, and are intended to show roughly the limits of the proposition of utilising waste heat.

As regards proposition II. for utilising waste heat there are very few mines indeed where the heat of the exhaust steam of winders and non-condensing engines and the circulating water of condensing engines could be utilised as assumed without going to very great expense and providing complicated systems of control and regulation so that the successful operation of the various plants would not be interfered with.

The whole details of such a system would have to be most carefully worked out, because in warm weather the water and solutions might become too hot for the successful operation in mill or cyanide works or for the purposes of condensing plant. Again, it must not be assumed that this so-called waste heat is all wasted because engineers in these days are making strenuous endeavours after efficiency and the prevention of waste of any kind. Exhaust feed water heaters are being installed on the exhaust end of winding engines to heat the feed water for the boilers with resulting economy in coal.

The subject of warming mill water is not a new one. Various plants were installed on the Rand years ago and are still working, in which the mill water is made to pass through the mill engine condenser, and some mill managers have laid many charges against warm mill water. The amalgam gets either too hard or too soft or sticks too tightly to the plates or is washed off, or the warm water interferes with the proper dressing of the plates, or in some way affects the amalgamating process ; what really does happen I have never fully enquired into, but my impression is that it largely depends upon the mill manager's troubles, which are promptly accounted for by "warm water."

The experienced engineer is not looking for troubles and further complications of plant, although he is at all times ready and willing to work out the practical details of any scheme which has for its object the betterment of conditions of working or increased efficiency on the mines in conjunction with either the miner or metallurgist.

In the case of the particular mine for which I had made preliminary calculations and estimates of cost for a system of utilising waste heat for slimes plant solutions, the cost was so great and the arrangement of the plant so awkward, involving extra pumping of solutions, and therefore raising the working cost so much that it was not considered to be worth the expense. Warm water throughout the plant would also mean increased water losses by evaporation, etc. As Mr. Salkinson also points out this question of warming solutions is not of equal importance for different plants. Where treatment capacity is ample (as it should be for efficient work), the practical utility of warming solutions is problematical, but in an undersized plant or when slimes particularly refractory as regards settlement are to be handled, the matter rests on a different basis. The whole question is, of course, a matter of cost. If the extra cost of heating the solutions is more than compensated for by increased total extraction, whether by increased capacity of the plant or by lower gold contents in the residues then it should pay.

It is unfortunate that in the two comparative results given by Mr. Salkinson for November, 1905, to April, 1906, without heat, and November, 1906, to April, 1907, with heat, the assay values of the slimes treated were not the same, for although this shews 1·7 per cent. higher ratio of dissolved gold with heat, the undissolved gold is practically the same in both cases. The assay values of slimes treated with warm solution are 1·898 dwt., as against 1·544 dwt., and one naturally looks to higher extraction with higher grade ore to work on. I should like to ask Mr. Salkinson what were the comparative results obtained with and without extra heat during the cold months, May to August, inclusive, as against the warm weather periods, November to April, which he quotes.

The problem appears to me not quite so simple and inexpensive looking as one would gather from the author's description when he says—". . . . . the cost of the process consists in the installation of an old boiler, or other suitable type of condenser, a pump for the circulation of solution, and the necessary piping."

There is no doubt, however, that the author has proved in the most practical manner the advantages of warm solutions for hastening the settlement of slimes and increasing the efficiency or treatment capacity of the plant with which he was working.

Mr. Jas. Lea: The author has pointed out in a very practical manner the influence of temperature on the settlement of slime. It is a subject that has very often been touched upon in the past proceedings of our Society, but few have realised the enormous advantage, and increase in the capacity of a plant, to be derived by the judicious application of heat.

It has been customary, for some considerable time, on the mine where I am, to take a quantity of hot water from the cooling pond to use as a supply for the tube mill spitzlutte. The hot water thus introduced into the mill water is sufficient to maintain the temperature at an average of 21° C. This is the average evening temperature, but I find the morning temperature generally 1° or 2° C. lower, the difference probably being due to the sun's heat during the day.

Some three months ago I thought it would be possible to get a little more work out of the tube mills if the spitzlutte had a better water supply than was at that time obtainable from the cooling pond, the alternative source being the mill water service. Before having this changed over I went into the times of settlement on the slime collecting plant, as this would be the first part of the slime plant to be affected by the change. The charges were settling in 8 hours from the time the stream was stopped running into the collectors, and I estimated we could easily get through the work if the time increased to 10 hours. This seemed to me to be a very liberal allowance. I did not think the change from hot to cold water would make this difference, so accordingly made the alteration. I found the temperature of the water dropped consistently 1° C. per day, and when it reached 18° C. my allowance of 2 hours per charge, or an increase of 25 per cent. in the time allowed for settlement, was considerably exceeded. On the fifth day when the difference in temperature was 5° C., the same size charges that had settled in 8 hours were taking over 12 hours. Needless to say I immediately had the hot water connected up again. In all, the hot water was off for 6 days, and when replaced, it took another 6 days to get back to normal conditions again. I should say that the increased water supply at the time was for two $\frac{3}{4}$ in. nozzles, but at the same time 20 stamps were hung up, so that the bad settlement was in no way due to an increased volume of pulp. Additional lime failed to compensate for the loss of heat.

No difficulty was experienced with the first and second treatment vats. This was probably due to the enormous amount of solution in circulation on the slime plant, and the cold water not having been on long enough to materially reduce the temperature of the mass, although the temperature of the solutions is considerably lower than the mill water.

It seems to me that the first place to start utilising waste heat is on the mill water. By so doing the whole plant from the collectors down is benefited, and there is quite an appreciable increase in the temperature of the sand and slime, which in turn help to maintain the temperatures of the solutions. This, however, would only be applicable on mines where there is a comparatively small quantity of mill water in circulation, and fresh water is taken into the plant to make up for losses by evaporation, etc. It would be out of the question on a mine where the mill water service is taken direct from a huge dam, into which the water from the plant was returned, as the loss of temperature would be too great on a large quantity.

I consider Mr. Salkinson's estimate of an increase of 20 per cent. in the capacity of a plant by heating the water and solution is very moderate in view of my very short experience on returning to cold water.

NOTES ON THE ESTIMATION OF CAUSTIC LIME.

(Read at August Meeting, 1907.)

By Edw. H. Croghan (Member).

DISCUSSION.

Mr. Andrew F. Crosse: I should like to make a few remarks on the excellent paper, lately read by my friend, Mr. Croghan, as I have been very intimately connected with analytical questions on the Rand for many years past, and I was the first to propose the use of the sugar process for the estimation of caustic lime. I wish to refer to Mr. Croghan's paragraph commencing "Available lime," which is the weakest paragraph in a most excellent paper. From the cyanide manager's point of view, taking a bag, or a ton, or any other quantity you like, for all practical purposes in cyanide operations, the value of that amount of lime depends upon the amount of soluble caustic lime, or CaO, which can be obtained therefrom. A gramme of finely powdered lime (but this must, of course, be a fair sample of the bulk) shaken up in a litre of boiled distilled water, would give perfectly satisfactory results, but I advise the addition of sugar in order to facilitate the solution of the lime.

I must thank Mr. Croghan for his valuable analyses of various samples of lime, and all that I can say is that in this and other questions of a like nature, a recognised standard of procedure or analysis should be arranged, in order to determine the commercial value on a recognised basis between buyer and seller.

There is one point which has never been seriously considered in this country. In England the lime is usually burned from amorphous rock, such as the well-known chalk, etc., in this country, however, the purest carbonate of lime is a highly crystalline deposit, dissolved out in most cases from the dolomite and re-deposited in caves and cavities. Now this crystalline limestone is very much more difficult to treat. If the lime-burners would use a little more scientific knowledge, they would find that even in crystalline limestone the CaO and the CO_2 would be more easily separated, as the chief impurity in local white lime, as far as my experience goes, is $CaCO_3$, proving incomplete calcination.

This gives me the opportunity which I have long desired, that is, to say something about the use of magnesia, MgO. Magnesia is practically insoluble in water, and therefore is inapplicable as a neutralising agent in cyanide solutions. Furthermore, cyanide of magnesium is, if it exists, a very unstable salt, and I most authoritively condemn all statements as to the possibility of using magnesia instead of lime, as a protective agency for cyanide solutions.

To my mind, the chief value of lime is its property of rendering the slime flocculent. Even cyanide of calcium is very unstable, and I am certain that anything like an excess of this salt is not advisable in working cyanide solution. Cyanide of calcium, if it were not so unstable, would be a cheaper material than cyanide of sodium, as $Ca(CN)_2$ contains a higher percentage of available cyanogen than even cyanide of sodium, but it is a very unstable compound: however, as Rudyard Kipling says, "this is another story," and it would require a paper to itself.

Will you allow me to wind up my remarks with an anecdote? A few years ago I was discussing, with the representative of one of the most important lime producers on the Rand, the advisability of slaking the lime instead of preserving it in an unslaked condition. I told him that slaked lime, properly protected, would remain good for many years, quoting an instance to which he replied, "slaked lime had been found in the Pyramids in Egypt, prepared over 3,000 years ago, and was still quite good!!"

The President: It is a pleasure to have our old friend, Mr. Crosse, with us again, and I would say to him that we often think of him, but we would think of him even more frequently if he would only contribute more to our discussions.

NOTES ON FEEDERS, WITH A DESCRIPTION OF A NEW DRIVING DEVICE.

(*Read at August Meeting, 1907.*)

By D. J. Pepler (Member).

DISCUSSION.

Mr. W. Beaver: There is only one portion of this paper with which I thoroughly agree, namely, that an increased stamp duty is very desirable from every point of view.

The device the author has brought to the notice of our Society is so similar to one in use at several mills on these fields, that I take exception to its being termed new. I cannot for a moment assume the author knows nothing of the ancient device I am about to describe, because one, if not several, was at work at the Rose Deep when Mr. Pepler took charge of that mill, some 18 months or two years ago.

Briefly, the difference is this, the friction plate, formerly used on the old Challenge feeder with the arm removed, is used in precisely the same manner as the author's driving wheel, with the two flanges; in every other detail the device is identical and works to perfection. Apparently he has wilfully overlooked the economical aspect of the method in order to make the device new, and call it "Pepler's operative device and Challenge feeder." There is positively no reason for the unnecessary expenditure for new wheels, the old friction plate being reliable in every way. The scheme is an excellent one I admit, but for one reason only, and that is because it requires much less attention than the three-pawl gear.

I have failed to locate any direct increase in my stamp duty after running this device for three years, and the only conclusion I can arrive at why his duty increased in consequence of the new device is, his three-pawl gear must have been in a shocking bad way.

The author also states that he has proved the device a saver of stem breakages to the tune of 40 per cent. I cannot imagine how he did it in so short a time, as six months hardly gives the scheme an opportunity of proving its worth, or otherwise, in that direction. Many things have to be considered. The character of the metal of which each stem is made is liable to differ considerably, so much so that some new stems might run months before breaking while others would break after running weeks, or even days; also the dies and the manner they might wear, are apt to differ somewhat, and so cause stems to break. Height of drop, weight o stamps, diameter of pulleys, state of guides, all these must be taken into account in a test of this kind, which I claim is impossible in a short run of six months.

Then again, he has compared this device with almost the oldest on these fields, the three-pawl gear; how about comparing it with some of the later ones, for instance, Smart's driving device, which I consider the best on these fields? This is an idea Mr. Smart introduced some time in 1898; there quickly followed a crop of similar gears, using the same idea, but applied in slightly different ways.

He goes on to say, that a reliable feeder will obviate the breaking of cams and cam-shafts to a great extent. To my notion the feeder plays a very small part indeed in breaking cams and shafts, always provided the stamps have the proper length of drop and are not camming.

He also states that the log kept of stem breakages shows a saving of 50 per cent. in favour of the new device, then further on, computing for the whole mill, the breakages were 40 per cent. less. Why this difference? If the trial was a fair test, and shows 50 per cent. less breakages, surely it should show the same figure throughout the mill. The author might reduce it still further in his reply, and bring it to something reasonable.

The first new device was put in on trial at the Ferreira Deep battery twenty months ago; allowing six months for experimental purposes, leaves fourteen months; in that time only three quarters of the mill has been equipped with the new device. Truly the word "hustle" must be unknown there; most "live" men when they get a good thing, know it, and get it working with the least delay possible.

The proceedings then terminated.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Method for Analysis of Gold-Silver Bullion*—"The rapid method here described was adopted for the approximate analysis of bullion produced by the Mijnbouw Maatschappij Redjang Lebong, Sumatra. It could, no doubt, be applied with slight modifications in other cases. It is possible to make all the determinations on the same weighed portion of bullion, but it is preferable to make an independent assay of the gold and silver on a separate portion.

The elements to be determined are:—Gold, silver, selenium, lead, copper, iron, zinc.

Gold.

Dissolve 1 gm. of the bullion drillings in a conical flask with two successive washings (25 c.c. each) of 75 per cent. nitric acid, boiling each time, diluting with an equal volume of water, and decanting after again boiling. Finally wash the residue once or twice by decantation with hot water. Treat all the liquid poured off as described under silver. Fill up the flask containing the undissolved residue, invert into a porcelain crucible, drain off the water, dry, ignite and weigh. This residue will in general be nearly pure gold, but it may be examined by dissolving in aqua regia, diluting, allowing to stand for some time for settlement of any insoluble matter, consisting of very small quantities of chloride of silver, carbon and silica. After carefully decanting the dissolved gold and washing with distilled water, the residue is collected, dried at a moderate temperature, and weighed. The weight so found is deducted from the total weight of residue insoluble in nitric acid to obtain the true weight of gold. The minute quantity of silver present is, perhaps, best determined by wrapping this residue (insoluble in aqua regia) in lead foil and cupelling.

Silver.

The united liquids poured off in the previous nitric acid treatment, together with the water washes, are heated to boiling in a flask, *A*, and mixed with about 5 c.c. of concentrated hydrochloric acid. Shake well until the AgCl begins to settle clear. Pour off the bulk of the liquid into another flask, *B*, and heat this to boiling. When the small quantity of suspended AgCl in *B* has settled clear, pour the liquid through a filter into another flask or beaker, *C*. Wash the bulk of the AgCl in *A* 5 or 6 times by decantation with boiling water, pouring off the liquid through the filter each time, and retaining the precipitate as much as possible in the flask. See that the filtrate passes through quite clear. Treat all the liquid collected in *C*, as described under selenium.

The silver may now be determined in various ways. It was found with this bullion that tolerably accurate results could be obtained by the method of G. Denigès, carried out as follows:—Place the flask *A* containing the bulk of AgCl under the funnel containing the filter used as detailed above. Measure out a quantity of a solution of cyanide of potassium or sodium, containing a little alkali, and of known cyanide strength, say, about 1 per cent The quantity of cyanide solution taken must be more than sufficient to dissolve the whole of the AgCl according to the reaction:—

$$AgCl + 2KCN = KAg(CN)_2 + KCl$$

Run this through the funnel into flask *A*; agitate thoroughly without diluting. When the bulk of the precipitate has dissolved, which requires some time, decant the liquid through the same filter, collecting in flask, *B*. Dissolve the small residue of AgCl in *A* by means of a few c.c. of concentrated ammonia; when quite dissolved, dilute and pass through the same filter into *B*, finally washing once or twice with water. To the filtrate add 10 c.c. of 1 per cent. potassium iodide, and immediately titrate the residual cyanide with standard silver nitrate until a permanent yellowish turbidity remains. Deduct the amount of $AgNO_3$ used from the $AgNO_3$ equivalent of the total quantity of cyanide added. The difference is the amount corresponding to the KCN converted into $KAg(CN)_2$, from which the amount of silver may be calculated:—

1 c.c. standard $AgNO_3$ (as generally used) = 0·00828 gm. Ag.

1 gm. Ag requires 1·208 gm. KCN.

The silver may be determined much more rapidly and conveniently by Volhard's method, but in this case either a smaller quantity of the bullion must be taken (say, 0·1 or 0·2 gm.) or the solution must be made up to a definite volume and an aliquot part taken. In using this method it was generally found convenient to add a slight excess of standard thiocyanate and titrate back with a standard silver solution of equivalent strength. The liquid after titration by Volhard's method cannot, of course, be used for the determination of iron, and the estimation of the other elements present would probably be inconvenient. This estimation is, therefore, always made on a separate portion.

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* See this *Journal*, vol. vii., June, 1907, p. 416.

In my experience at Redjang Lebong, the results by Volhard's method were generally lower, and those by Denigès' method higher than the fire assay, the latter being made with the usual corrections as described in a former paper.*

SELENIUM.

Evaporate the filtrate collected in *C* to dryness on the water bath, adding 0·5 gm. sodium chloride. At the finish add about 10 c.c. of concentrated hydrochloric acid and again evaporate to dryness, repeating this operation if the residue is considerable. Dissolve in the least quantity of hot dilute HCl: add 5 to 10 c.c. of concentrated HCl and boil thoroughly. Hold a piece of paper moistened with KI at the mouth of the flask and continue to boil, with occasional additions of fresh HCl, until the paper no longer shows any discoloration, which indicates that the last traces of HNO_3 are expelled. Then dilute to 50 c.c. or 100 c.c., according to the amount of dissolved salts, boil and filter if necessary, washing well with boiling water. Examine the small insoluble residue on the filter for silica and traces of silver chloride. Treat the liquid as follows:—

Pass a current of purified SO_2 gas into the hot liquid for some time until the latter smells strongly. Boil till the red precipitate turns to a dark violet, almost black: allow to settle, covered, for a few minutes and filter while still hot, preferably collecting the precipitate on a paper previously dried at 100° C. and weighed. Wash thoroughly with boiling water, adding a little SO_2 water and HCl from time to time. See that the filtrate is perfectly clear and gives no further reaction with SO_2 on boiling. A crystalline precipitate of lead chloride will probably separate on cooling, but this is of no consequence. Treat the filtrate as described under lead.

This method of separating lead and selenium was found to be much more satisfactory than that described by H. Rose (*Handbuch der Anal. Chem.*, vol. ii.), in which the lead is precipitated first with sulphuric acid. The liquid cannot be boiled with this acid without loss of selenium, and unless heated to white fumes the precipitation of lead is incomplete. The addition of alcohol also introduces difficulties and complications. So far as I am aware the method of separation here detailed is quite new.

The estimation of the selenium in the precipitate may be made in the ordinary way, by drying in an air oven at 100° C. and weighing the paper and precipitate together, inclosed in a weighing tube or bottle. This is probably the safest method, but it is always advisable to examine the selenium for small quantities of gold, which may have been dissolved in the nitric acid treatment. This may be determined by wrapping the weighed filter paper and contents in a considerable excess of lead foil, scorifying and cupelling.

As the drying and weighing at 100° C. is somewhat tedious, attempts were made to utilise the volumetric method, depending on the reaction of selenious acid with potassium iodide:—

$$4KI + 4HCl + H_2SeO_3 = Se + 4I + KCl + 3H_2O$$

Under suitable conditions this was found to give good results with pure solutions of selenious acid, but it was found difficult to convert the selenium into selenious acid without loss. The method tried was as follows:—

Wash the precipitate off the paper as much as possible into a porcelain evaporating dish; drain the water back through the filter into a flask. Add to the precipitate in the dish 10 c.c. of 75 per cent. nitric acid and 0·5 gm. of NaCl. Add the water previously poured off and evaporate on the water bath. Digest the paper in another dish with hot 25 per cent. nitric acid, adding the liquid to that in the first dish when the residue adhering to the paper is completely dissolved. Treat the evaporated residue with HCl, as already described, till all HNO_3 is expelled, cool and add 0·5 to 1 gm. of potassium iodide and titrate the iodine liberated with $\frac{N}{100}$ thiosulphate (2·48 gm. $Na_2S_2O_3.5H_2O$ per litre), with addition of starch indicator near the finish, until the bluish-black tint changes to orange red. It is essential to standardise on a selenious acid solution of known strength and not to depend on the theoretical standard calculated from the equation.

A further check may be made by using the resulting liquid containing the red selenium in suspension for a colorimetric test. Checks are made by taking measured volumes of the standard Na_2SeO_3 solution, adding HCl + KI, and removing the iodine by addition of thiosulphate until the resulting mixture has the same tint as the liquid derived from the bullion.

LEAD.

To the filtrate from the selenium precipitate add 10 c.c. concentrated sulphuric acid and evaporate to a small bulk. Finally boil in a conical flask with constant agitation over a spirit flame until copious white fumes are given off. Cool, dilute to about 50 c.c. and add 10 c.c. of alcohol. When thoroughly settled (preferably after standing over night), filter, wash with dilute and finally with absolute alcohol. Treat the filtrate as described under copper.

Wash the precipitate off the filter into a clean flask, add ammonia, then acetic acid in slight excess; boil till all is dissolved, or if any residue remains, filter this off, wash with hot dilute ammonium acetate, and examine. Pass the hot clear filtrate through the original filter paper, first moistening the latter with dilute ammonia. Wash with hot water and a few drops of acetic acid. Again add 5 to 10 c.c. of concentrated H_2SO_4 and boil to white fumes.

The lead sulphate should now be pure and the lead may be estimated by the ordinary gravimetric or volumetric methods. After settling for some hours, with addition of dilute alcohol, it may be collected on a weighed perforated crucible containing an asbestos filter, ignited gently and weighed as $PbSO_4$. The quantity is generally too small for satisfactory results by volumetric methods.

$$PbSO_4 \times 0·683 = Pb$$

COPPER.

Boil the filtrate from the $PbSO_4$ thoroughly to expel alcohol. Pass a current of H_2S gas into the hot liquid till it is saturated. Filter, wash with H_2S water; treat filtrate as described under iron. Wash the precipitate off the paper as much as possible into a clean flask *A*. When settled, pour back the liquid from *A* through the same paper into another flask, *B*, retaining the precipitate in *A*. Add to *A* 5 c.c. concentrated HNO_3, warm gently till dissolved, adding a little $KClO_3$ if necessary. Add the liquid poured off into *B* and again heat until only yellow particles of sulphur remain. Filter through the same paper, dry and burn the paper and add the ash. Again heat to boiling. Filter and estimate the copper volumetrically by the iodide method, or when only a little is present, add ammonia in slight excess, filter, and determine copper by the colorimetric method.

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* *Engineering and Mining Journal*, Dec. 8, 1906, p. 1057.

The liquid after the latter test may be examined for cadmium by adding KCN till colourless, then H_2S.

IRON.

Transfer the filtrate from the copper sulphide to an evaporating basin. Boil to expel the excess of H_2S. Add HNO_3, boil, then add ammonia in slight excess. Boil, filter; treat the filtrate as described under zinc. The iron is generally determined by colorimetric test, as follows:—

Dissolve the precipitate by passing 10 to 15 c.c. of hot 25 per cent. HCl through the filter, collecting the liquid in a Nessler tube. Wash once or twice with water; cool the tube, add 5 c.c. strong (say, 5 per cent.) KCNS. In another tube place an equal amount of HCl and also 5 c.c. of KCNS. Dilute to nearly the same volume as the first solution and run in standard ferric chloride (1 c.c. = 0·1 mg. Fe) to the second tube until the tints are alike.

In rare cases when sufficient iron was present for a volumetric assay, the ferric hydrate was dissolved in dilute sulphuric acid, boiled with aluminium turnings till the iron was completely reduced to the ferrous condition, cooled and titrated with permanganate.

ZINC.

As practically no other metal was present ~~in~~ the filtrate from the iron, it would seem that zinc might be determined directly by acidulating with HCl and titrating with ferrocyanide at a suitable temperature. It was generally found, however, that the results were unsatisfactory, perhaps owing to the presence of nitrates, unless a separation was made. This was carried out by acidulating very slightly with sulphuric acid, and precipitating the warm solution (at about 70° C.) with H_2S.

The zinc sulphide is then easily filtered off and the zinc determined by any of the usual methods. Dissolve the ZnS in a slight excess of HCl. Boil, filter, dilute; heat in a porcelain basin to 70° C. Run in standard ferrocyanide till a drop taken out gives a distinct brown colour with a drop of uranium acetate on a white plate. Add 1 or 2 c.c. ferrocyanide beyond this point. Allow to stand in a moderately warm place for 10 minutes, then titrate back with standard $ZnCl_2$ until a drop no longer gives a colour with the uranium indicator. The standard solutions in use were each adjusted so that 1 c.c. = 0·0025 gm. Zn.

GENERAL REMARKS.

An analysis by this method can generally be carried out in two days. On the first day the gold, silver and selenium are determined and the lead precipitated as sulphate in the filtrate from the selenium. On the second day the lead, copper, iron and zinc are determined. The results obtained usually agreed satisfactorily, and left about 0·25 per cent. unaccounted for. This system of analysis is only applicable to the 'cyanide' bullion, 'mill' bullion containing too large a proportion of gold."—J. E. CLENNELL.—*Engineering and Mining Journal*, June 8, 1907, p. 1099. (G. H. S.)

THE DURABILITY OF COMMERCIAL LEAD.—"The leaden pans employed in concentrating sulphuric acid sometimes 'break down' and undergo extensive corrosion even when carefully used at fairly low temperatures. Sometimes the breakdown occurs suddenly with the evolution of hydrogen sulphide or sulphur-dioxide and the production of lead sulphate in quantity. At other times the action is slow and the gases are not evolved in appreciable quantity. Chemically pure lead is much less corroded than the impure metal. Of the metallic impurities copper, arsenic, and silver are least harmful, but bismuth, tin, aluminium, antimony, and zinc all render the metal very liable to acid corrosion. Physical condition plays an important part in the durability of lead even when the metal is exceptionally pure. The loose crystalline structure produced by casting the lead at too high a temperature is answerable for many cases of lead failure. The altered physical structure produced by rolling under great pressure also diminishes the resisting power of the metal, and when the deformation is very extensive this ill-effect may become permanent."—*Times Engineering Supplement*, July 31, 1907. (J. A. W.)

THE ATOMIC WEIGHT OF POTASSIUM.—In this research KBr was prepared (1) by the action of bromine on potassium oxalate (2) by evaporation of mixed solutions of NH_4Br and KOH, all the materials having been previously most carefully and exhaustively analysed and prepared. Then the ratio Ag : KBr was redetermined and found to be 100 : 110·319, from which K = 39·1143 if Ag = 107·93 and Br = 79·953. The ratio AgBr : KBr was found equal to 100·000 : 63·373, which gave K = 39·1135. By a similar series of experiments carried out on KCl, the number obtained was 39·114.—T. W. RICHARDS and E. MUELLER.—*Journal American Chemical Society*, vol. xxix., No. 5, p. 639, May, 1907. (J. A. W.)

TRANSMUTATION OF ELEMENTS.—"An article appears in the *Chemiker Zeitung* by Prof. Wilhelm Ostwald, the well-known chemist at the University of Leipzig, in which reference is made to Sir William Ramsay's discovery that elements heretofore regarded as unchangeable are capable of transmutation. Prof. Ostwald alludes to it as *the greatest scientific achievement since the discovery of the practicability of applying the electric dynamo to mechanics*, and he describes at length the process by which Sir William Ramsay converts radium into helium and produces neon, krypton, lithium, and sodium. He then says:—'When I visited Ramsay in London he demonstrated to me that he could produce lithium from copper by the action on a solution of sulphate of copper of the emanations of radium. After the copper had been extracted, by means of sulphuretted hydrogen, from the solution which had been in contact with the emanations of radium, a residue of lithium remained. Since then, as I have seen from the proofs of an article that Sir William Ramsay is about to publish, he has not only confirmed this discovery, but has added to it. He has proved that, should the emanations of radium, alone or mixed with hydrogen, be left in a vessel, after a time helium will be produced. In the event of the emanations coming into contact with water, instead of helium, neon with slight traces of helium is the result. Then by dissolving in the solution a heavy metal (the experiments were carried out with the nitrate of silver and the sulphate of copper) xenon or krypton is produced. Other substances are also present, but no exact definition of their character has been obtained, owing to the infinitesimal quantities in which they occur. Sodium and calcium have been observed among them, but the latter may possibly have come from the experiments having been made in glass vessels."—*Times Engineering Supplement*, August 21, 1907. (J. A. W.)

The Composition of Slag Cements.—"Some important investigations on the composition and physico-chemical aspects of the highly basic Portland cements, made from blast furnace slags, are dealt with by Professor Zulkowski, of Prague, in *Stahl und Eisen.* It is pointed out that in slag cements the proportion of calcium in combination is always much lower than in ordinary Portland cements, and it is asserted that he has been able to prove that this is due to the formation of dicalcium metasilicates, in which it is possible that an excess of lime may be present in actual solid solution. As distinct from other investigators, he is of the opinion that in no case can tricalcic silicates be formed, and that both in ordinary Portland cement as also in slag cement the silicates of lime and the silicates of alumina exist as double silicates."—*Times Engineering Supplement*, Aug. 7, 1907. (J. A. W.)

Reduction of Oxides and Sulphides with Metallic Calcium.—"The author referred to the well-known powerful reducing action of aluminium, as, for example, its use in the preparation of chromium, ferro-silicon, and other metals and alloys, and its use in the form of "Thermit" for welding purposes. He finds that metallic calcium is a still more powerful reducing agent than aluminium. For example, when molecular proportions of aluminium and ferric oxide are mixed together and ignited by means of a fuse of aluminium and barium peroxide, intense reaction ensues, and continues until all the oxygen has been removed from the ferric oxide, and aluminium oxide and metallic iron produced in its stead. When metallic calcium in the form of fine turnings is mixed with ferric oxide and ignited in a similar manner, the reaction is so intense that the mixture is in large part ejected from the crucible. The reaction can be brought under control by mixing 30 to 40 per cent. of calcium fluoride, or 10 to 20 per cent. of calcium oxide with the contents of the crucible. Boron can be obtained by lighting a mixture of boron trioxide with the calculated quantity of calcium, and 5 to 10 per cent. of calcium oxide. If no calcium oxide is added the reaction is too violent. It is, however, very difficult entirely to free the boron from calcium even after standing over hydrochloric acid for weeks; the boron invariably contains considerable quantities of calcium, which is probably present as calcium boride. When borax is carefully dried and mixed with calcium, and the mixture ignited by a fuse, the reaction completes itself without undue violence. On treating the reaction mixture with hydrochloric acid, brown amorphous boron is obtained in almost theoretical quantity: the product, however, always contains calcium, but can be used for preparing boron chloride and similar compounds. Rather anomalous results were obtained when sand (silicon dioxide) was ignited with calcium. Unless considerably less than the theoretical amount of calcium was used, not silicon but probably calcium silicide was obtained, because when the reaction mixture is placed in hydrochloric acid, large quantities of a spontaneously inflammable gas are given off: this gas is probably silicon hydride. The product remaining behind is silicic acid and calcium chloride. The results were the same when a mixture of aluminium and calcium was employed.

Attention was also drawn to the difficulty experienced in causing silica in the form of fine sand to react with metallic aluminium which was in the form of a rough powder. A similar difficulty was met with in the case of boric anhydride and aluminium. Galena does not yield metallic lead and calcium sulphide, but a greyish mass which gives off sulphuretted hydrogen when acted upon with acids, lead in the form of a salt being left in solution. Red phosphorus and calcium unite with explosive violence. Sulphur and calcium also react with great vigour. Manganese dioxide and calcium require to be mixed with 30 to 40 per cent. of calcium fluoride, when a regulus of manganese can be obtained, otherwise the reaction is too violent. This is also the case with chromium oxide. When the experiments were first tried it was found that the reaction mixture always stuck to the crucible, and it was invariably necessary to break the crucible in order to obtain the product. Subsequently the difficulty was got over by lining the crucible with a paste made from slaked lime and a strong solution of sodium silicate (water glass); in the case of boron, the solution used was generally borax. The crucible and lining were thoroughly dried, and were then ready for use. With crucibles so prepared there was never any difficulty in getting out the reaction product in the form of a lump without breaking the crucible."—Dr. F. Mollwo Perkin.—*Iron and Coal Trades' Review*, from Faraday Society, June 14, 1907, p. 2,124. (A. McA. J.)

Phase Relations of Sodium Carbonate and Water.—The experiments described in this paper concern the phase relations of sodium carbonate between 25° and 45°, an interval which includes all the known transition points of the hydrates. Three definite compounds containing 10 H_2O, 7 H_2O, and 1 H_2O, were obtained, and the transition points found were 32° for 10 H_2O – 7 H_2O, 32·96° for 10 H_2O – 1 H_2O, and 35·37° for 7 H_2O – 1 H_2O. The authors state that the transition points, and solubility determinations are now in good accord.—R. C. Wells and D. J. McAdam, Jnr.—*Journal American Chemical Society*, vol. xxix., No. 5, p. 721, May, 1907. (J. A. W.)

Errors in Volhard's Method for Halogens in the Case of Chlorides.*—The method consists in acidifying the given solution with HNO_3, adding a ferric salt as an indicator, precipitating the chloride with a measured but excessive amount of a standard $AgNO_3$ solution and without filtering out the AgCl determining the excess of $AgNO_3$ by titration with standard NH_4CNS solution. An extensive experience of this method showed that in its present form this method was inaccurate and historical investigation proved that others had had the same experience.

Drechsel, in 1877, found that the reaction:—$Fe(CNS)_3 + 3AgCl = FeCl_3 + 3AgCNS$ took place, and hence recommended filtering out the AgCl before titration. Sutton recommends this, but other analytical authorities titrate without filtration. The authors therefore set out to determine if the velocity of the reaction was as slow as was assumed by most chemists. Experiments soon showed that it was very rapid, and also the error of the experiment, if no filtering were made, was found to vary up to 2·9 per cent., whereas with filtering the relative error was found 0·17 per cent. Hence they state that filtration is indispensable, although it diminishes somewhat the rapidity of the method. With bromides and iodides similar experiments were carried out and it was found that filtration was not necessary here.—M. A. Rosanoff and A. E. Hill.—*Journal of the American Chemical Society*, vol. xxix., 3, March, 1907, p. 269. (J. A. W.)

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* See this *Journal*, p. 300, vol. vii., March, 1907.

METALLURGY.

A Cheap Form of Cyanide Plant.—"During the past two or three years the small mining propositions in Rhodesia have contributed materially to the gold output, and it is estimated that at present over one-third of the total declared value is thus obtained.

The majority of these propositions are let on tribute, while others are worked by the owners. The reduction plant, as a rule, consists of gravity or Tremain steam stamps, though a few Huntington mills, and possibly a Kinkead mill or two, are also in use.

While the recovery by amalgamation is fairly satisfactory, the values left in the tailings are too high to be thrown away. At the same time the operator is often unable or unwilling to spend any great amount on a cyanide plant. To meet this case, a light and portable form of plant has been put on the market and is largely in use throughout Southern Rhodesia. These plants give good results, last well, and are simple and convenient in operation.

The following is a copy of the contract and specification for one of these plants which has come under the direct notice of the writer during the past few months, and it is put forward in the hope that it may prove of assistance, or at least of interest, to members.

Copy of Contract.—This contract to comprise every operation necessary for the complete equipment of a cyanide plant, including all materials and labour, except the items of labour and material hereinafter specified.

The delivery of the plant to be free at the nearest railway siding. Mine owner to provide transport between siding and mine.

All work to be done under supervision of the mine owner (or his representative) and to his satisfaction.

The plant to consist of the following :—

A.—Five leaching vats, 10 ft. diameter × 4 ft. 2 in. deep.

B.—Two solution tanks, 8 ft. diameter × 6 ft. deep.

C.—Two extractor boxes of seven compartments each, made of iron, compartments 15 in. × 15 in. × 15 in., with 3 in. baffle-board partitions.

D.—One steam cyanide pump, Knowles duplex type.

E.—All necessary pipes, valves, fittings, and hose necessary for the complete equipment from and to tailing and solution tanks and extractor boxes.

Water connections and steam connections, including stop valves.

F.—All necessary gratings, matting, jute, cloth, Manila rope and packing.

G.—Distributor tray for receiving pipes and hose connections from leaching vats, with partition and two outlets to slimes catchers.

H.—Two slimes catchers, each 16 in. × 16 in. × 16 in., with baffle board.

Details of Plant.—

A. and *B.*—Leaching vats and solution tanks.

These to be of the dimensions mentioned and to be constructed of 22 B. W. G. galvanised corrugated iron, riveted and soldered inside and outside. Tops of such vats to be stiffened with 1 in. × 1 in. × ¼ in. angle iron.

C.—Two extractor boxes.

These boxes to be of seven compartments each, 3 in. space between baffle board partitions, constructed of 22 B. W. G. galvanised flat iron, riveted and watertight.

(These extractor boxes are made of 22 B. W. G. iron, and not of wood, as the iron boxes are both cheaper and lighter and are not liable to leakage through warping or cracking.)

D.—Steam pump.

Knowles duplex type with 1½ in. delivery and 2 in. suction.

E.—Valves, pipes and fittings.

All solution pipe 1¼ in., leaching pipe 1 in. Return to solution tanks from pump 1¼ in. Return from boxes to solution tanks 1½ in. To supply only connections and valves to water supply and steam to pump.

To include all necessary flanges, bends, tees, plugs, valves and foot-valves necessary for the proper equipment of such pipe line.

F.—Gratings and accessories.

All necessary gratings for leaching tanks constructed of 1 in. × 1 in. strips on 3 in. × 1½ in. bearers, with perfect sweep sides built in two sections, and neatly covered with approved cocoa matting and jute cloth turned over at edge and caulked with Manila rope.

G.—Tray with partition, 2 ft. 6 in. × 3 ft. 3 in. × 6 in.

To receive leaching valves and hose connections.

Tray to be made of 22 B. W. G. galvanised flat iron with outlets to slimes catchers.

H.—Slimes catchers.

Two boxes 16 in. × 16 in. × 16 in., each with baffle board across centre.

All constructed of 22 B. W. G. galvanised flat iron, and to have outlet to extractor box.

General.—All galvanised iron work to have one good coat of Stockholm tar applied hot.

The entire plant to be supplied, erected, and handed over in running order for the sum of £300 sterling.

The plant actually cost a little more than £300. A wattle and daub hut was erected over the extractor boxes at a cost of £5.

A charging platform and gangway, pine planks and native timber struts, cost approximately £12 10s., and transport from a siding two miles away was £2 10s.

The total weight of the plant was 9,000 lb., and the £300 quoted may be divided as follows :—

5 leaching tanks @ £20 ...	£100	0	0
2 solution tanks @ £15 ...	30	0	0
2 extractor boxes @ £12 10s. ...	25	0	0
Matting and rope	13	0	0
5 gratings @ £7	35	0	0
Piping	40	0	0
Cyanide pump	26	10	0
Excavations, foundations, erection and railway fares and carriage	30	10	0
Total	£300	0	0

The first of these plants was erected in March, 1905, and is at present showing no signs of wearing out, although it has been running constantly.

It was paid for out of the first two months profit, and others have done the same. This plant was constructed of 24 B. W. G. iron instead of the 22 gauge iron of the later plants.

The following are details of some of the installations :—

No. 1.—Capacity 400 tons per month.
- 6 leaching vats 10 ft. × 4 ft.
- 9 solution tanks 15 ft. × 6 ft. (those are extra large).
- 2 extractor boxes (wooden), each with seven compartments, 15 in. × 15 in. × 15 in.
- Excavations not included. Plant and erection £300.

No. 2.—Capacity 800 tons per month.
6 leaching vats 15 ft. × 4 ft.
2 leaching tanks 12 ft. 6 in. × 6 ft.
2 extractor boxes.
Supplied and erected for £570.

The cost of running a 400 ton cyanide plant is approximately 4s. 6d. per ton, as under:—

	s.	d.	
European labour (half charged to mill, half to cyanide)	1	9	per ton.
Native labour	1	0	,,
Cyanide (1 lb.)		10	,,
Zinc (⅓ lb.)		3	,,
Lime		1	,,
Stores and general		7	,,
Total	4	6	per ton"

— CHAS. HUNTER. — Institution of Mining and Metallurgy. *Bulletin No. 34*, July 18, 1907, p. 21. (R. G. B.)

UTILISATION OF "BLACK SAND."*—"The author served under the U.S. Geological Survey in the black sand investigation at Portland, Oregon, for 2½ months in the summer of 1905. In general the conclusions were as follows:—

That if we sized the sands on 8 mm., 2 mm. and ½ mm. screens, this would give us four sizes. The first upon 8 mm. was in every instance rejected as having no value. However, it could have been jigged or hand-picked if that appeared necessary. The second size (which passed through the 8 mm. and rested on the 2 mm.) was jigged in certain cases, which took out quite a lot of gold, and we found some of the other heavy minerals. We did not go further with this concentrate, but commercially it would be in condition to treat further if its value so indicated. The third size (which passed through 2 mm. and rested on ½ mm.) was run on a Wilfley table, and this in case of certain hydraulic clean-up sands yielded much free gold, together with other heavy minerals available for after separation. The portion which went through the ½ mm. screen was in every case treated on Wilfley (or other riffle table, of which we had with us the Pinder, the Woodbury and the Christenson tables), and this treatment yielded a concentrate of heavy mineral and, if so desired, a middling product, and tailings which were let go to waste.

The treatment was mainly that devised by Henry E. Wood, of Denver, and consisted in drying the concentrates and subjecting them to several successive treatments of Wetherill magnets, taking out the strongly magnetic minerals with a weak current in the first passage, and successively weaker magnetic grains with stronger currents in the following passages. We found, for example, that magnetite was lifted out with 0·2 ampere, chromite and ilmenite were lifted out with 0·8 ampere, that garnet was lifted out with 1·75 amperes. Hypersthene was lifted out with 1·9 amperes; this being a useless mineral, was simply thrown away. Monazite was lifted out by 3 amperes, and zircon was left behind with the gold. The platinum came largely with the chromite and garnet, and the iridosmium came mostly with the zircon.

We also added two points, both of which seemed to us important, to the list of known methods of handling these minerals. First, we found that a 2 gallon bottle revolved slowly endwise on a flange on the end of a shaft would extract in one hour from a kilogram of black sand all of the fine native gold when the sand was treated wet with 50 gr. of concentrated sodium amalgam. This assay of sand proved so valuable that some of our Wilfley table runs checked up to a point of accuracy where the sum of the products gave 99 per cent. of the value fed. Secondly, we proved that this concentrated sodium amalgam would amalgamate platinum and the platinum metals as freely as gold, and that as soon as the sodium was spent by oxidation in contact with water, the platinum would come out from the amalgam as quickly as it went in, and the mercury or the gold amalgam in it would have no more to do with the platinum grains than if it were so much sand. This seemed to point to a means of recovering platinum from concentrated black sands if the quantity of platinum were sufficient and the cost of sodium did not prove too high; and also the separation of platinum from gold."—ROBT. H. RICHARDS.—*Engineering and Mining Journal*, June 29, 1907, p. 1,251. (G. H. S.)

MAGNETIC CONCENTRATION OF IRON ORES.—"Many low grade and impure iron ores may be commercially utilised for economic blast furnace practice and for the production of good quality pig iron by means of the Gröndal process. Among these ores, which have hitherto proven of little or no economic use, are:—magnetites of all kinds, including magnetic iron sands, containing as low as 25 per cent. iron; magnetites containing a high percentage of phosphorus and copper not chemically combined with the iron; purple ore and pyrites residues: nearly all iron ores containing a high percentage of sulphur, and flue dust. The Gröndal process consists of the following method of treatment:—Dry crushing to about ½ in. cubes; wet crushing in Gröndal ball mill to from 10 to 100 mesh, as may be necessary; passing of ground pulp through Gröndal magnetic separator for a removal of non-magnetic particles; and pressing of the concentrates thus obtained into briquettes, which are heated in the Gröndal briquetting kiln—the briquettes becoming hard, porous, easily reducible ferric oxide, containing a minimum amount of sulphur. The cost of separation-concentration, in treating a 40 per cent. magnetic iron ore, resultant in concentrates of 63 per cent., varies from 30 c. to 45 c. per ton of concentrates; the cost of briquetting varies, with the size of the plant, from 45 c. to 85 c. per ton. In Sweden, Norway and Finland, the process has a wide application, and the successful results of treatment are well shown in the following figures:—Ore containing 58·96 per cent. iron, 0·036 per cent. sulphur and 1·29 per cent. phosphorus, yielding concentrate of 72·38 per cent. iron, 0·003 per cent. sulphur and 0·005 per cent. phosphorus, and briquettes of 69·49 per cent. iron, 5·502 per cent.* sulphur and ·006 per cent. phosphorus; crude ore running 36·43 per cent. iron, 0·021 per cent. sulphur and 0·318 per cent. phosphorus, made into a concentrate of 71·76 per cent. iron, 0·015 per cent. sulphur and 0·008 per cent. phosphorus. Ore from Cornwall, Penna. running 30·65 per cent. iron, 0·260 per cent. phosphorus, yielding a concentrate of 69·59 per cent. iron, 0·132 per cent. sulphur and 0·007 per cent. phosphorus. Ore from Cornwall, Penna., running 30·65 per cent. iron, 1·603 per cent. sulphur and 0·012 per cent. phosphorus, was made into concentrates of 69·95 per cent. iron, 0·036 per cent. sulphur and 0·003 per cent. phosphorus. Ore from Cornwall, Penna., running 30·65 per cent. iron, 1·603 per cent. sulphur and 0·012 per cent. phosphorus, was made into concentrates of 69·95 per cent. iron, 0·036 sulphur and 0·003 per cent. phosphorus, and into briquettes of 69·90 per cent. iron, 0·010 per cent. sulphur and 0·005 per cent. phosphorus. At the Fitzgerald and Bennie laboratories, Niagara Falls, Ontario, there is a Grondal testing plant operative, and here crude titaniferous ore, carrying 16·57 per cent. TiO_2, was made into concentrates of 4·21 per cent. TiO_2 and middlings of 19·48 per cent. TiO_2—these concentrates being obtained by only one passage through the separator, and it is probable that finer grinding and repassing through the separator would have given even more satisfactory results.

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* See this *Journal*, vol. vii., June, 1906, p. 418.

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These interesting data we have abstracted from a paper by P. McN. Bennie, presented before the Canadian Mining Institute."—*Mining Reporter*, May 23, 1907, p. 473. (J. Y.)

METALLURGY OF ALUMINIUM IN 1906.—A. Gawalowski has studied the formation of aluminium carbonate and has prepared a carbonate at ordinary atmospheric pressure, which is insoluble in water and contains 8 to 9 per cent. alumina. At 8 atmospheres pressure a poly-carbonate soluble in water is obtained which decomposes on reduction of pressure, the bearing of these facts on the deposition of bauxite being apparent.

F. R. Pyne has studied the melting points of cryolite-alumina mixtures with the following results:

Cryolite.	Alumina.	Melting Point.	Cryolite.	Alumina.	Melting Point.
100	...	1,000° C.	93	7	982
97	3	974	92	8	992
96	4	960	90	10	980
95	5	915	85	15	994
94	6	960	80	20	1,015

—J. W. RICHARDS.—*Engineering and Mining Journal*, June 8, 1907. (G. H. S.)

BORON STEELS.—"The preparation of these alloys offers no difficulty as ferro-boron is now a product of the electric furnace. The micro-structure of boron steels shows the existence in the material of perlite corresponding to the amount of carbon present. The ferrite gives with picric acid a coloration indicating the presence of a solid solution of ferro-boron: it also exhibits a series of hard granules which appear to consist of iron borocarbide. This borocarbide, which renders the steel very brittle, passes into solution when the metal is tempered at 850° C., provided that the proportion of boron is low, but a portion remains undissolved when the amount of this element present is about 0·8 per cent. It is probably these tempered boron steels which will find employment in the arts."—*Times Engineering Supplement*, July 31, 1907. (J. A. W.)

THE BUTTERS FILTER.*—"So much has recently been written about the respective merits of the Butters and other filters and so few actual data have been given that it would interest the mining public to get the working costs, as well as the metallurgical results achieved by the Butters filter during the period of continuous running.

The article by Mr. F. L. Bosqui, in your issue of February 2, covers the ground so well from my point of view that I shall simply endorse it as written and make no further comment. I would point out that my figures relate to one year's continuous running, and that the filter in question has been steadily working for more than eighteen months. Nevertheless, it is in as good condition to-day as when it was first erected.

The high cost of freighting between the Coast and Copala tends to increase the cost of maintenance. The following is the summary:—

FOR YEAR ENDING JANUARY 31, 1907.

Total tonnage filtered	24,066 tons
Average filter-residues during year	gold 9 cents.,
,, ,, ,, ,,	silver, 0·98 oz.
Average charge per filter of 94 leaves	20·02 tons
Number of charges	1,202
Average time of operating one charge	1 hr. 36 min.

COST OF MAINTENANCE.

Acid (HCl) for washing filter leaves one year	$1,596·00
Repairs to filter	2·69
Repairs to 4-in. centrifugal pumps	49·60
Repairs to vacuum pump	3·50
Total	$1,651·79

COST OF POWER.

Vacuum pump (4 h.p.) 1,622 hr., at 17 cents. per hr.	$110·29
Centrifugal pump for elevating excess pulp and solutions, 8 h.p., 801 hr.	108·93
Centrifugal pump for elevating the filtrate, 8 h.p., 365 hr.	49·64
Excess slime agitator, 2·5 h.p., 550 hr. ...	23·37
Total	$292·23

COST OF LABOUR.

One man at $45 per month, one year ...	$540·00
One boy at $12 ,, ,, ,, ,, ...	144·00
Extra help, washing frames, etc.	32·00
Total	716·00

Yearly cost of maintenance ...	$1,651·79
,, ,, ,, power	292·23
,, ,, ,, labour	716·00
Total yearly cost... ...	$2,660·02
Cost per ton of slime	$0·1105

The capacity of the filter is 200 tons per day, but it has never been taxed to even 50 per cent. of its capability; in fact, the unit-charge is now 26·5 tons and the results are quite as good as with the smaller tonnage.

In conclusion, I would say that we have gone on record with a complete sheet of operations and it would be interesting to see similar data published from the other filters on the market. Mr. Bosqui's suggestion is a practical one."—JAMES FRIER, Copala, Mexico.—*Mining and Scientific Press*, April 6, 1907, p. 431. (K. L. G.)

CARBON AND PHOSPHORUS IN STEEL.—J. E. Stead has performed experiments for the elucidation of the banding of carbon and phosphorus in rolled steel such as rails, and explains the results observed as follows:

In cooling the steel containing carbon and phosphorus, when crossing the critical range of temperature $A_3 - A_1$, the excess ferrite separating out of the austenite carries with it in undue proportion most of the phosphorus, leaving, of course, the carbon with the austenite, which, after passing the A_1 points reverts to perlite; the perlite areas being surrounded by a network of excess ferrite.

Consequently, when, as a result of rolling, the ferrite and perlite are stretched out into bands, the carbon content and phosphorus content of the steel will also appear to vary in bands, and a sort of incompatibility of carbon and phosphorus thus indicated, also prevents the diffusion of carbon and phosphorus when the steel is again heated beyond A_1, leaving them still distributed in bands on cooling. H. M. HOWE.—*Engineering and Mining Journal*, June 8, 1907. (G. H. S.)

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* See this *Journal*, vol. vii., May, 1907, p. 377.

TELLURIUM IN GOLD BARS. "The presence of tellurium in the bullion produced by treatment of the ores of the Kalgoorlie field is generally recognised. The following table shows the results obtained by analysis of gold bars produced in the treatment plant of the Great Boulder Perseverance Gold Mining Co., Ltd. :—

Description of Bullion.	Base parts per 1,000.	Tellurium parts per 1,000.	Remarks.
Cyanide bullion...	49·0	12·2	Before utilisation of lead acetate in agitation vats.
,, ,,	29·5	2·84	With ,, ,, ,,
,, ,,	27·0	1·39	,, ,, ,, ,,
,, ,,	36·8	11·67	Bullion produced after 15 days' run. No lead acetate used.
,, ,,	22·6	Trace	Bullion produced at end of following 15 days. Lead acetate used during period.
Amalgam bullion	27·0	0·34	Amalgam obtained from grinding pans
Bullion from cyanide retreatment of roasted ore residues ...	30·2	Nil	From lakeside treatment plant.

Comparison of the tellurium contents of the various cyanide bullions points to the fact that the lead acetate, added as a corrective for sulphides to the pulp in the agitation vats, precipitates also any tellurium from solution, in which this element exists combined most probably as potassium or sodium telluride."—*Journal of Chamber of Mines*, West Australia, Nov., 1906, p. 461. (H. A. W.)

VELOCITY OF FALLING GALENA AND QUARTZ IN WATER. — Rittinger's formula for the velocity of particles falling in water is shown to apply only above a certain critical velocity.

In the case of quartz this critical velocity is 28 mm. per second, and for galena 63 mm. per second, the corresponding critical diameters of particles being 0·20 and 0·13 mm. respectively.

Below this critical velocity the law is expressed by the formula

$$V = K(\delta - 1)D^2$$

and above by Rittinger's

$$V = C\sqrt{D(\delta - 1)}$$

$K = 424$ for quartz and 631 for galena.

$C = 87$ for quartz and 100 for galena.

$D =$ diameter of particle in millimetres.

$\delta =$ density.

Other minerals than quartz and galena may be dealt with on the assumption (approximately correct) that the velocity of such grains will be proportional to their specific gravity. — R. H. RICHARDS. — *Bi-monthly Bulletin of the American Institute of Mining Engineers*, May, 1907. (G. H. S.)

ELECTROLYTIC LEAD REFINING.* — Lead bullion treated (at the works of the Consolidated Mining and Smelting Co. of Canada, Ltd.), contains :—Silver, 100 oz. ; gold, 1 oz. ; arsenic, 0·2 per cent. ; antimony, 0·8 per cent. ; copper, 0·25 per cent., and is cast into anodes 31¼ × 26 in. by 1½ in. thick, weight 300 lb. each, with lugs at two corners.

Cathodes are thin sheets of pure lead, 26 × 36 in. $\frac{3}{16}$ in. thick, supported from copper bars ; at the end of the operation they are washed free from slime and melted down in a 50-ton melting pot and cast into bars, containing 99·9962 per cent. lead, 0·0014 per cent. silver, no gold, no arsenic, and 0·0006 per cent. antimony.

The electrolyte is a solution of lead in hydrofluosilicic acid, silicon fluoride running between 8 and 11 per cent., and lead from 4½ to 5½ per cent. : it is kept at about 30° C. and about ½ lb. glue per ton of lead refined is added to prevent formation of lead crystals.

The refining tanks are 240 in number, 84½ × 30 in. by 41 in. deep, arranged in double rows and in the cascade system, each tank 3 in. lower than the preceding one, so that the electrolyte circulates by gravity, passing from tank to tank through hard rubber pipes, which draws it from 7 in. above the bottom of the upper tank and discharges it at the top of the next, finally passing it through settling tanks to the lower pump tanks whence it is raised to the upper tanks ready for circulation, by a lift pump made of copper.

The electrodes in each tank are in parallel, and the tanks are in series on a 110-volt 4,000 amp. circuit. Current density is 16 amp. per sq. ft. for anodes and 14 amp. for cathodes. Each tank holds 20 anodes and 21 cathodes.

The tanks are unloaded every eight days, two cathodes having been used to the one anode. Short circuits are tested for by voltmeter every 12 hours. Poor contacts between electrodes and bus bars are frequent.

The smoother the cathode, the purer the deposited lead, as irregularities catch particles of floating slime, which in turn cause formation of lumps on the surface.

The scrap lead (undissolved anodes) is freed from slime by leather edged scrapers and remelted.

Every 24 days the tanks are completely emptied and the slime recovered in the settling tanks.

Slimes analysed as follows :—Water, 14·5 per cent. ; gold, 0·1 per cent. ; silver, 17·1 per cent. ; copper, 9·5 per cent. ; antimony, 25·91 per cent. ; arsenic, 5·96 per cent. ; lead, 14·5 per cent.

The slimes are transported to washing tubs by means of copper cars, washed with hot water and transferred to filter tank in which the filter bed consists of a sheet of hard rubber with many small perforations, filtration being aided by steam suction. The filtered slimes next go on drying cars into the dryer, a sheet iron hood 6 ft. wide by 13 ft. 10 in. long.

The dried slimes are mixed with soda and melted down in a small water-jacketted reverberatory lined with magnesia brick. Arsenic, antimony, copper and lead are oxidised and slagged, skimmed off and returned to the lead blast furnaces. The doré bullion obtained runs 950 to 960 fine and is cast into bars weighing about 38 lb., and with two holes through them to facilitate parting.

Parting by sulphuric acid is adopted with subsequent precipitation of silver on copper ; copper being recovered as sulphate by crystallisation.

The gold residue is treated with hot water and finally with nitric acid, washed again, dried in a safe provided with an electric heater inside, and finally melted with borax to bullion 998 fine. The precipitated silver is washed and charged into a hopper feeding into a reverberatory with hexagonal hearth taking about 80,000 oz.; when melted it is covered with charcoal and ladled into bars.—A. G. WOLF.—*Bulletin Colorado School of Mines*, vol. iv., pp. 11—30. (G. H. S.)

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* See this *Journal*, vol. vii., April, 1907, p. 347.

MINING.

ROYAL COMMISSION ON MINES AND BREATHING APPARATUS.—"The first report of the Royal Commission on mines has recently been issued, and contains recommendations as to the provision and use of breathing apparatus under conditions where dependence upon the air of the mine is either hazardous or impossible. The consideration of the question of watering in collieries to prevent or mitigate dust explosions, upon which a great deal of evidence has been taken and published, is postponed certainly till the autumn, and in face of the opposition to the proposal from colliery managers (due more, perhaps, to the wide-spread incredulity as to the reality of this oft-demonstrated peril, which we have so often noted, than to any inherent impossibility of solution), it is improbable that any further progress can be made till an experimental gallery—such as that at Frameries in Belgium, or at Dux in Bohemia can be constructed, from which further data may be obtained. As regards breathing appliances the subject has received so much notice in our columns that any introductory observations are unnecessary, and we will pass at once to consider the data and recommendations furnished by the current report. The entire Blue Book issued includes besides the report itself, an account, written by Dr. A. E. Boycott, of the Lister Institute, of a series of important investigations carried out for the most part under laboratory conditions with the best known makes of breathing appliances, and a review of the conclusions of a French commission which studied the question last year under the stimulus of the Courrières disaster, and of the legislation consequent thereon.

Considering that our own Commission's report follows the conclusions of the scientists who made the experiments which are the subject of Dr. Boycott's report, it is perhaps most convenient to briefly consider these before passing the Commission's own findings in review. From the point of view of application, the instruments studied were intended (*a*) for short and light service, (*b*) for arduous work over longer periods; and from the point of view of construction, fell into three groups—(1) Those in which highly compressed oxygen is supplied to the subject from steel cylinders which he carries, and the expired CO_2 absorbed by being passed over potash. (2) Those in which, by passing the expired air through a cylinder containing peroxide of sodium and potassium, not only is the CO_2 absorbed but oxygen regenerated, by which the necessity of carrying an oxygen reservoir is obviated. (3) Those in which respirable air is supplied by the evaporation of liquid air from a reservoir carried by the user. Besides these types, there is, of course, the one familiar to the public from diving practice, where air is pumped into a helmet by means of a tube of greater or less length. With this type Dr. Boycott's report does not deal, among other reasons, probably because of the detailed experiments recently carried out by the Admiralty with this class of apparatus, the data from which should be published ere long. The pneumatophores actually tested were the following:—'Shamrock,' 'Fleuss,' 'Dräger,' 'Weg,' of Type (1), 'Pneumatogen,' of Type (2), and 'Aerolithe' of Type (3). The experiments are described with great minuteness, but space forbids us doing more than tabulating the chief features recorded of each, which are as follows:—

Pneumatogen I.* was found to supply sufficient oxygen to admit of resting for 1¼ hours and walking gently for 30 minutes, and it is described as 'admirably adopted for the purpose for which it was designed—*i.e.*, to enable workmen to escape from the face to the shaft.' Of the larger apparatus type (b), the verdict is that 'it cannot be recommended in its present form, in spite of its great superiority in weight.' The objections to this type, however, appear to be mainly mechanical, and if the frothing of the sodium-potassium peroxide, which after a while prevents the free action of the CO_2 on the cartridges, could be overcome so that the supply of oxygen were satisfactory, the regenerative principle of the machine offers great advantages. The Shamrock apparatus is stated to be 'seriously defective on account of the imperfect absorption of the CO_2 though otherwise very well designed.' This machine is also criticised in details, in respect of the oxygen cylinders not being independently controlled, of their being too thin, and of the gauge glass being invisible to the user. The aerolithe apparatus is considered impracticable apart from the production of liquid air on a commercial scale, while in operation there is no good indication of the amount of oxygen remaining for use—a circumstance unnerving to the operator and attended with the possibility of suddenly fatal results where the constitution of the wearer does not give him premonitory symptoms of a deficiency of oxygen. The Dräger apparatus is stated to 'afford an abundant supply of oxygen, and the CO_2 is absorbed very thoroughly. On the other hand, the apparatus is heavy and badly hung, so that it is uncomfortable and tiring in use and liable to catch.' The gauge also is not visible to the wearer, and the thickness of the oxygen cylinder is criticised, as are minor points in construction. The Weg apparatus, which is the one designed and made by Mr. W. E. Garforth, was the only one tried under practically service conditions in the experimental chamber at Altofts. It is criticised on the ground that the helmet will fit only the individual for whom it has been made. This (apart from the possibly slight additional expense) is a serious objection in some cases, as if great heat has to be encountered one individual may not be able to stand the conditions for long and the pneumatophore cannot then be utilised by a fresh operator. It is for this reason perhaps, and because the inventor hopes to improve it, that the Weg apparatus has not yet been placed on the market, as no other serious exception is taken to it. The Fleuss machine, which is the prototype of all those relying on compressed oxygen, is described as 'well made and comfortable to wear,' while the alterations in the improved pattern 'have effected most substantial improvements.' The lower oxygen capacity as compared with the German types was regarded as conditioned by a desirable margin of safety in the strength of the oxygen cylinders. As this is the first detailed examination of the best known types of pneumataphores carried out by physiological experts under ideal conditions for scientific observation and air analyses that we know of, the results seem to us to merit detailed examination. Dr. Boycott points out that from the physiological point of view two conditions have to be kept in view:—first, that while there is no considerable physiological advantage in increasing the percentage of oxygen above 20 per cent., it is undesirable that it should fall much below that figure; while, secondly, as regards carbonic acid, the percentage should not rise much above 1 per cent.

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* See this *Journal*, vol. vi., Dec., 1905, pp. 197-9.

Turning now to the Report of the Commission, we find from the appendices that the use of breathing appliances has been made compulsory in Austria since 1897, in Russia since 1902, in Silesia and in France on April 15 of the present year. The Committee, however, while expressing the opinion that 'the results of the experiments are satisfactory,' and that 'the question demands the serious attention of the industry,' is averse from any compulsory regulations, at least at present. Not only may the appliances themselves be greatly improved, but the whole question of organisation and training suitable for differently situated mines, which, now that we are assured of possessing various efficient appliances, is, perhaps, the most important practical question, has to be thought out and tested. The main point at present is that it should be realised both by manufacturers and mine managements that there is an important if limited branch of mine equipment which has hitherto been neglected, but which it will be necessary to provide in future, if not under regulation then in the interests of the safety of the mine and its employees, especially with the possibility of the provisions of the new Employers' Liability Act being held applicable to engagements extending outside the United Kingdom. Owing to the impulse in favour of the use of pneumatophores having been largely given by the Courrières disaster there is perhaps a disposition among metal miners to regard the question as peculiarly one for coal miners. Such, however, on reflection will be seen not to be the case. In the present stage of development the practical use of the pneumatophore will, we fear, be found much more in the exploration of foul workings and the location and extinction of fires, than in rescue work after an extensive explosion. Instances and arguments could be adduced in favour of this view but space does not permit. These services are scarcely, if at all, less necessary in metal than in coal mines. The disastrous fires at Broken Hill last year might have been prevented by the possession of a pneumatophore service, and the enquiries we have received relative to the adequacy of various apparatus indicate that in some metal mining centres, at any rate, the matter is receiving consideration. Perhaps a general conversion of sentiment would be most complete and speedy if insurance offices interested in mining business were to realise the extent to which the security of mines from fire and its consequence—suffocation—is increased where proper appliances and organisation exist for dealing with irrespirable conditions underground."—*London Mining Journal*, July 20, 1907. p. 91. (J. Y.)

Modern Applications of Electricity to Mines.—"The importance of increasing the application of power to reduce labour costs renders the use of electricity a matter of interest to all mine owners, as no other system has equal flexibility in distribution, or is so well adapted to meet every form of drive above or below ground.

Economics. — The principal economies resulting from the use of electricity are through:—(1) Reduced stand-by charges; (2) small loss in transmission; (3) high efficiency of the electric motor; (4) the lower capital cost of producing and transmitting energy. The saving is specially noticeable where the workings are spread over a large area, as the whole of the steam plant can be concentrated at one point.

The cost of electric power varies over wide limits and depends upon—(1) The size of the undertaking; (2) capital outlay (influenced by locality); (3) cost of fuel and circulating water (if water power is not available); (4) load factor.

Where electric driving has been adopted on an extensive scale, the load factor at the power house is between 30 and 40 per cent. This high load factor makes the fuel cost important. With the cheap fuel available at collieries, the cost of the coal only accounts for one-quarter of the total cost at the power house; whereas, when fuel is purchased, its cost may amount to more than one-half of the total; thus the advantage to be gained in applying electric driving in mines, where fuel has to be purchased, is much greater than the economy obtained in colliery working.

Advantages for Underground Work.—The difficulties of ventilation in collieries, even when no gases have to be dealt with, render the application of steam difficult, and until electricity was applied compressed air was the only available agent for the inner workings.

In consequence of the high capital and running costs of compressed air, the haulage gears were generally steam driven, being placed on the surface or at the bottom of the intake shaft.

Principal Applications.—While electric motors are now used for every form of colliery drive, the most important applications in England have been for haulage. Owing to the intermittent character of this work, the percentage saving by remodelling existing haulages is greater than that obtained in modifying any other part of a colliery equipment. Next in importance to haulage is the use of electricity for pumping, which has received great impetus through the development of the high-lift centrifugal pump. Owing to the comparatively large powers, the foregoing uses attract special attention, but the use of electricity for detailed purposes, while already of importance, will attract even greater attention in the future. Great progress has been made in electric coal-cutting, which not only reduces the cost of winning a ton of coal and increases the percentage of large coal won, but allows the thinner seams to be profitably worked, thus increasing the coal reserves.

The possibilities of electric drilling are now coming to the front, and it is in the extension of the use of electricity in detail that the future reduction in mining costs are to be looked for.

Electric Winding.—While there is a substantial saving in the running costs, owing to the low cost of fuel available at collieries, this saving is at present counterbalanced by the interest charges due to high capital cost of electric winding plant; in consequence this is the last part of a colliery plant to receive attention in conversion from steam to electric working. Where the hours of winding are long, or the cost of fuel high, a substantial saving can now be made by adding electric winding, the whole power required being obtained from a single power house.

Haulage.—In addition to the lower power costs the first advantage gained with electrically driven haulage is the freedom in choice of position and speed; and second, the decreased wear and tear on the ropes and road through the steadier drive.

In collieries it is seldom possible to consider the use of electric locomotives, owing to the difficulty of sparkless collection of current, but where mines are sufficiently dry and gradients not excessive, a considerable gain in efficiency and maintenance will result in operating by electric locomotives in place of main and tail haulage.

Pumping.—The comparatively high speed at which it was necessary to run the motors in order to keep down the capital cost was a disadvantage in the first pumping applications when three-throw pumps were driven by ropes or gearing, but the combination of the electric motor with the high lift centrifugal pump has resulted in a great advance on previous methods of pumping.

The high speed and purely rotary motion has much reduced the capital cost and the space occupied compared with any previous system."—C. PRATT SPARKS at Engineering Conference.—*Iron and Coal Trades Review*, June 28, 1907, p. 2,288. (A. R.)

ANKYLOSTOMIASIS IN SCOTLAND.—"Mr. Maclaren, Inspector of Mines for the East of Scotland District, in his annual report, says:—'Information reached me that two miners from the south of Edinburgh had been treated for this disease, and I at once made investigation and found that it was the case. The facts are that the men had worked in the gold mines in India—one for about two years, and one for fully a year. Both came back to this country in August, ill and suffering from the disease; the one least affected started in a mine in Edinburghshire, and worked from the 25th August until 2nd October, when he was informed by the manager that if he had the disease he must leave the mine, which he did, and two days later he went to the Edinburgh Royal Infirmary, where he was treated. After the medical authorities had satisfied themselves that he was no longer affected with the worm, he was discharged cured on the 27th November. The other man, who did not start in any mine, was admitted to the infirmary on 31st August, and at the end of the year was not cured, but it is stated he is recovering. The section of the mine where the man worked was dry and had a moderate temperature, and while employed as a miner he was cutting a 'trouble,' and only he and his drawer worked in the section. He stated that only on one occasion had he discharged fæces, and it was covered up in the waste. The men who came in contact with him have since been examined and found to be free from the symptoms of the disease and in satisfactory health.'"—*London Mining Journal*, June 15, 1907, p. 807. (J. Y.)

HANDLING OF DYNAMITE.—"At a meeting of the mine inspectors for the anthracite region of Pennsylvania, held recently at Hazleton, the following letter on the subject of thawing, handling, and general use of dynamite was drawn up:—

We recommend, as a body of mine inspectors, having in view the preservation of life and property, that the manufacturers of high explosives mark plainly upon each box, and stick, the percentage of the principal ingredients contained therein, so that the purchaser can see the quality of powder* he is using. In view of the numerous accidents occurring throughout the anthracite coal fields, due in a great measure to the fact that the user of the explosive has not the proper knowledge of same, the lack of which, in our opinion, falsely impresses him with the belief that he is handling, or using, a grade of powder having 20 per cent. nitro-glycerine, when he is actually using a 40 per cent. powder, the indifferent handling of which tends to increase the number of accidents.

It is our opinion that with a proper knowledge of the grade being used, there will be less recklessness.

We also recommend that the strongest cap procurable should be used for the exploding of dynamite, and that date of manufacture be on every box of dynamite. We further recommend that the caps be placed in the box with the open end down, to avoid sparks from lamp or pipe falling into the cap when box is opened, and these should not be kept in the same building or box as the dynamite.

The thawing of dynamite received considerable consideration by us, and the conclusion which we arrived at was, a simple, safe, and practical solution to the question would be difficult to obtain, and after a lengthy discussion on the subject, advise that the safest method of thawing dynamite (especially in the mine) was by the use of warm manure, or the thawing kettle, as recommended by the manufacturers of high explosives, being two water-tight kettles, one smaller than the other, the cartridges to be placed in the smaller kettle, and the space between the two filled with hot water of from 130° to 140° F., the kettle being fitted with a cover to retain the heat. Under no condition, however, should the kettle be placed over the fire to heat the water. When more heat is required, empty the cold and refill with warm water."—*Mines and Minerals*, July, 1907, p. 574. (A. McA. J.)

MISCELLANEOUS.

WATER IN THE EARTH.*—"Recently before the Royal Institution Prof. John W. Gregory presented a view of the condition of water at temperatures above its critical point. It is evidently his opinion that the elements of water are not combined with each other above the critical temperature of liquefaction, for he says:—'Water, although its constituents may come from vast depths within the interior, is limited to a depth of perhaps only six or seven times the depth of existing mines. The lower limit is due to the internal heat of the globe. . . . Now, water cannot exist at a temperature higher than its critical point, 687° F. . . .' Again, 'At depths below about 37,000 ft. the temperature would be above the critical point of water, which therefore could not exist as such. Its elements would be given forth as separate gases from the slowly cooling mixture; the gases would rise, and having passed into a zone with a temperature below the critical would combine to form water.'

The Professor has, a chemical expert says, confused the critical temperature of liquefaction, 358° C.,† with the temperature of dissociation of water, which is probably about 2,500° C. The only change which water undergoes at its critical temperature is the loss of its surface where it is in contact with a gas or vapour, no matter how great the pressure may be. If the view set forth in the quoted extract were true, the maximum temperature to be obtained by the combustion of hydrogen and oxygen would be 687° F., which is near the boiling point of mercury."—*Science and Art of Mining*, July 27, 1907, p. 591. (T. L. C.)

A NEW HIGH-POWER GAS LAMP.—"The great progress in artificial illumination during the last decade has been accompanied not only by superior efficiency but also by increased intensity of light. Since the introduction of the electric arc lamp into the sphere of lighting vigorous efforts have been made to augment the illumination obtainable from oil and gas in order to approach as nearly as possible to the intensity of the electric arc light.

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* *Anglicè*, explosive. [Ed. Com.]

* See this *Journal* vol. vii., Nov. 1906, p. 153.

† These temperatures are somewhat inconsistent. [Ed. Com.]

High illuminating power can, however, only be produced when the body emitting the light is raised to a very elevated temperature. In general the tendency of modern efforts has been to achieve this object by the use of intensified gas as well as by enlarged mantle surface.

Whilst the well-known Welsbach burner, in conjunction with the best mantles, gives a light of about 100 to 120 Hefner candles, and as regards efficiency is about as cheap again as the newest one watt metallic filament lamps, a far greater lighting effect can be produced when the gas is ignited under high pressure in special burners. The efficiency then amounts to 1 Hefner candle for about ·75 to ·8 litre of gas per hour, and great light centres of above 1,000 candle-power can be obtained, such as are worthy to be compared with electric arc lights. In recent years a number of more or less ingenious contrivances have been invented based on the compression system. These usually work up to a pressure of 100 mm. of mercury, whilst formerly every endeavour was made to keep the pressure much lower than this. In order to produce the required pressure special arrangements were necessary, and a number of drawbacks have prevented this system from coming into general use.

Quite recently, however, a new high-power lamp has been invented which, when connected to an ordinary gas installation, produces the increased pressure automatically. Lucas, the inventor of this lamp, has previously rendered great services in the sphere of lighting, and he has now succeeded in constructing a lamp of considerably greater illuminating power than those previously introduced. It is a well-known fact that the illuminating properties of the incandescent burner largely depend upon the proportion and the uniformity of the mixture of gas and air. The ordinary Bunsen burner does not comply with these conditions, since with normal gas pressure, the flame is only furnished with about half the quantity of air necessary for complete combustion. This proportion should be one part of gas to 2·8 parts of air, and therefore the flame, in order to obtain the requisite amount of air to complete the combustion, must draw supplies from the atmosphere in the immediate neighbourhood of the burner. In the Welsbach light this lateral absorption of air takes place all round the mantle, and is associated with a definite lowering of temperature, so that the gas consumed in the burner does not produce the same heating effect which can be obtained with a more perfect mixture of gas and air in the burner itself.

In 1895-6 Denayrouze endeavoured to produce an improvement in these respects by arranging in the mixing tube of the burner a small fan driven by an electric motor, and in this way he was able to produce a much more intense light centre. These high-power burners have been introduced in France to a somewhat large extent, but they are only intended for medium light centres of several hundred candles. Whilst the Denayrouze burner must derive the current necessary for driving the small motor from a special source of electricity, Lucas in his new high-power lamp has removed this drawback by arranging in the upper part of the lamp a thermopile, which is actuated by the heat contained in the spent gases. The current necessary for driving the motor in the lower part of the lamp is thus generated by the lamp itself in burning.

In construction the new lamp is exceedingly simple. A thin tube suspended from the main pipe serves to feed the gas into two mixing tubes, and between these lies the armature of a small electric motor. In order as much as possible to avoid friction the vertical shaft of the armature runs on a point in a brass tube filled with oil. A fan fixed on the same shaft draws in air from below, and forces the intimate mixture of gas and air through two lateral conducting tubes into a small chamber from which the mixture passes to the burner orifice itself. The hot gases rising from the flame meet the sections of a thermopile, arranged in the upper part of the lamp. This source of electricity supplies the current for driving the small motor, and the wires terminate in two minute brushes of silver wire on the commutator of the motor.

Thermopiles have been known for a long time. In the present case a special form of construction was necessary which should be above all things durable. Lucas made use of two metallic alloys, one consisting of copper and aluminium, the other composed of copper and nickel. These alloys, which are extremely heat resisting, are employed in the form of metal strips, which are soldered hard at the heated end, and soft at the cool end. The thermopile is arranged in the form of a disc, and is capable of being easily replaced. Equally ingenious are the small motor and fan, so that these parts are not likely to get out of order, even after long usage.

On starting the lamp it burns at first under normal gas pressure with a flame of little illuminating power. Presently the hot gases in the chimney heat up the inner junctions of the thermopile, and generate the current, so that the motor soon begins to drive the fan at a high speed. After half a minute the fan already rotates at a speed of 2,000 revolutions per minute and the lamp emits a light of from 1,100 to 1,200 Hefner candles, consuming about 900 litres of gas per hour.

Prof. Wedding has subjected this new high-power lamp to a searching examination and has found that it gives far better results than the existing pressure gas lamps. It possesses the great advantage that it is independent of any machine, and can be attached to any ordinary gas installation and set into action at once.

In the tests of the lamp it has been found that after 1,460 hours' use it still continues to work efficiently, and the oiling only requires to be renewed after 500 to 600 hours' burning. The incandescent mantle lasted 70 hours, which is about the same as in the well-known compressed gas lamps. The mantle holder lasts 300 hours. The manipulation of the lamp is simple.

The Lucas lamp must be regarded as an important step in advance in the sphere of high-power gas lighting. It affords a simple solution of a problem which has been under investigation for a number of years. Its utility in practice has been put to the requisite proof by a year of uninterrupted observations, so that it may now be said to be ready for general introduction." — Dr. C. Richard Bohm. — *Times Engineering Supplement*, June 12, 1907. (J. A. W.)

DECOMPOSITION OF CEMENT BY SEA WATER.—"M. H. le Chatelier states that all hydraulic binding agents, without exception, are capable of being decomposed by sea water, but that the rate of this action varies within wide limits. It becomes relatively slow when alumina is present only in small proportions. The addition of *puzzuolana*, especially of calcined clay and of trass of good quality, endows all hydraulic cements with considerably increased powers of resistance to the action of sea water. The most essential condition to render cements immune from decomposition in the sea is to reduce to the utmost the volume of water employed in gauging the cement, and it is here that the effect of *puzzuolana*, quite independent of its chemical action, becomes of value."—*Times Engineering Supplement*, July 31, 1907, p. 244. (J. A. W.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

PRACTICAL COAL MINING.—Divisional Volume II.—By Prof. W. S. BOULTON, B.Sc., F.G.S. 10 × 6½, vi. + 187 pp. Price 5s. 6d. net. (London: The Gresham Publishing Co.)

"The complete work is to embrace six volumes and is to be the combined work of several writers. The volume under review is the product of three of them, Prof. Louis contributing the larger share in a well-written article on shaft sinking which was begun in Volume I. He considers his subject under two heads—ordinary methods and special methods of shaft sinking. And if the matter strikes one as rather the outcome of a study of the literature of the subject than of direct practical experience, the processes are so complete and well described as largely to compensate for this deficiency, and examples culled from the pages of the journal *Glückauf*, of Reimer's work, and of the transactions of the Institution of Mining Engineers, and elsewhere, are put before the reader in a concise and profitable form.

Mr. Bulman's contribution, under section v., on 'Breaking Ground,' furnishes some useful information of a thoroughly practical nature relative to the cost of driving stone drifts, a very welcome departure which might be followed with advantage in other works on mining. On coal-cutting by machinery few people are better qualified to write than Mr. Bulman, as all who have perused the reports of the Coal-Cutting Committee of the North of England Institute of Mining Engineers must acknowledge. It was therefore to be expected that this subject would be, as it is, adequately treated. An oversight, however, is the complete omission, both from this and the preceding sections, of any account of power-driven drills, though hand-drilling occupies four pages and is illustrated by six figures in the text and one plate. No work on mining can be considered complete that does not devote some space to power-driven rock drills, and the proper place for their treatment would have been under 'Shaft Sinking' or 'Breaking Ground.'

'Methods of Working and Timbering,' which, after all, makes up the greater part of mining, occupy but 54 pages. The writer, Prof. Roberton, is in error in his supposition (p. 296) that the cleavage of coal is more pronounced in the hard steam coals of the North of England than elsewhere. As a matter of fact, it is just as marked in the softer house coal seams of Durham, and much more so in the steam coals of South Wales. Nor is he correct in his statement that the method known as Longwall is 'the outcome of the exhaustion of the thicker seams of coal in the North of England, which necessitated some practical ways being found of working the thinner seams with profit' (p. 226). Longwall working was practised in the Midlands and in South Wales long before it was introduced into the North of England, being originally known as the 'Shropshire' method of working. The volume is profusely illustrated, and many of the figures will be recognised as old friends. Those in the section on sinking are of most value. The type and 'get up' of the book are all that could be desired, and the comprehensive nature of the work renders it a valuable addition to the mining library."—*Times Engineering Supplement*, July 24, 1907. (J. A. W.)

SOLUBILITIES OF INORGANIC AND ORGANIC SUBSTANCES. A handbook of the most reliable quantitative solubility determinations. Re-calculated and compiled by ATHERTON SEIDELL. 12s. 6d. (London: Crosby Lockwood & Son.)

"The system adopted by the author—of publishing only those results which he judged to be best—is one to be highly commended, since a perusal of a mere collection of the immense mass of recorded observations, often carelessly made and published without proper confirmation, commonly misleads the inexperienced searcher. The investigator must, however, keep both Storer and Comey in his library as a further check, or as a vehicle for further search into any special question as to solubilities, and particularly for supplementing Seidell's work, which is avowedly less full than either of the earlier works, and is, we take it, intended mainly for the busy man, and covers only such solubilities as the average man is likely to require.

Salts of the rare elements are in many cases ignored, and in others treated very unceremoniously, but here again we are in general, though not full agreement with the author. To treat all equally would have meant an extremely bulky volume, heavy expense, and, more serious still, long delay, but we hope that Dr. Seidell may be able to amplify his work and include in his next edition much that it is at present ignored.

Such salts as uranium acetate, ammonium molybdate, and many others might well have been included, and one must always remember that many rare elements which were quite recently of academic interest only are to-day employed commercially, and are daily increasing in importance.

As a rule, it is better to refer to the original papers on questions of solubilities, etc., whenever possible, but when this is not essential the work of Dr. Seidell should save an immense amount of trouble. The method of selection adopted, as outlined in the preface, appears to be the best possible one, taking it for granted that typographical errors and mistakes in calculations have been elimated as far as is humanly possible."—*London Mining Journal*, Aug. 3, 1907, p. 152. (A. R.)

AFTER EARTHQUAKE AND FIRE. Large 8vo. 194 pp. San Francisco: *Mining and Scientific Press*, 1906.

This is a reprint of the articles which appeared in the *Mining and Scientific Press*, immediately after the earthquake and fire at San Francisco, April 18, 1906, and forms a record of extreme interest.

Three days after the catastrophe, the weekly issue appeared in the form of a single sheet, announcing the destruction of the whole of the plant, manuscript, library, etc., of the paper, and asking in dignified language, well worthy of being read and considered, for the continued support and goodwill of subscribers.

At the same time, the essentially American business instinct is well in evidence; the business manager announcing that a copy of the mailing list had been saved, that new offices had already been secured in Berkeley, and arrangements made for future printing, and that the next issue would afford the best medium for advertising very necessary changes of address.

The articles, numbering nearly forty in all, deal chiefly with the earthquake and subsequent conflagration, both receiving consideration from many points of view.

Possible causes are dealt with fully, as also are the lessons to be learned and applied in future; one of those being the prevention, as far as possible, of the use of dynamite by amateurs in preventing the spread of conflagrations.

The opinion is also expressed that if only the houses had been built on sound foundations, by honest workmanship and of good design, the damage suffered from the earthquake would have been but slight, except, perhaps, on made or soft ground, buildings on which always suffered the greatest damage.

Most of the damage was caused by fire, rendered possible by the centralisation, and, subsequent to the earthquake, consequent complete dislocation of the water supply.

The miscellaneous articles are all of interest to mining and metallurgical men; one notices another characteristically American statement—an address to American mining students—to the effect than an English engineer at a new mine always sees first to the building of a nice house to live in, and after that to the development of the mine.

The book is copiously and well illustrated, and generally excellently turned out. (G. H. S.)

THE PRINCIPLES OF COPPER SMELTING. By EDWARD DYER PETERS. Large 8vo. 612 pp.: New York and London, 1907.

This important volume, as stated by the author, deals with the "reasons why," instead of the "means how" in copper smelting, and hence is a welcome and very valuable adjunct to the same author's work on "Modern Copper Smelting."

The subject is dealt with in great detail; the author has taken considerable trouble to elucidate fully every point dealt with, by the consideration of various cases which may present themselves to the smelter, and also by the plentiful use of numerical examples, and has not stopped short of repetition and re-repetition, when by so doing the facts could be impressed upon his readers.

Furthermore, the student is never allowed to lose sight of the practical aspect of theoretical considerations, and is constantly reminded throughout the book that the main aim of the smelter is to earn dividends.

Though primarily intended for students who may be supposed to be well grounded in chemistry, Prof. Peters has, in order that the book might be readily intelligible to others possessing comparatively slight knowledge of the subject, written in such a manner that this end is attained, yet there can be but few, even in the highest branches of the profession, who could peruse it without gaining some advantage.

There is, in all probability, no other metal in the smelting of which progress has been more rapid than with copper. One reads here of blast furnaces now in operation, 87 ft. long by 56 in., smelting 3,000 tons of charge per day, in place of those of only two or three years back of 15 ft. by 56 in. with a smelting capacity of 400 tons per day.

Space forbids an extended notice of the contents, as no branch of this large subject is neglected; but special mention must be made of the section on "Pyrite Smelting," largely written by Mr. Robert Sticht, of Mount Lyell. In this the phenomena observed are correlated and explained, and knowledge of the subject brought up to date.

Other sections on "Chemistry of Smelting," "Roasting," "Slags," "Furnace Building," and "Thermo Chemistry," cannot fail to be of value to any concerned in other branches of metallurgy than copper smelting. (G. H. S.)

TIN DEPOSITS OF THE WORLD. By SYDNEY FAWNS. 2nd Edition. 8vo. No. 12. 304 pp.: *Mining Journal*, London, 1907.

All that was said of the first edition of this book (see this *Journal*, vol. vi., p. 182) holds good of the second, but its value is enhanced by the inclusion of a clearly written chapter on tin smelting, in which there seems to have been no important change during the last decade.

The book now contains, therefore, a complete account of the occurrence, mining, assaying, and metallurgy of tin ores, and brings together in reasonable compass all that is of any practical importance in connection with these subjects. (G. H. S.)

REPORT ON THE EXPERIMENTS MADE AT SAULTE STE. MARIE, ONTARIO, UNDER GOVERNMENT AUSPICES, IN THE SMELTING OF CANADIAN IRON ORES BY THE ELECTRO-THERMIC PROCESS.* By EUGENE HAANEL. 4to. 149 pp. and numerous plates. Ottawa, 1907.

The experiments were undertaken to determine the following points:—

1. Can magnetite, which is our chief ore, and which is to some extent a conductor of electricity, be successfully and economically smelted by the electro-thermic process?
2. Can iron ores with comparatively high sulphur content, but not containing manganese, be made into pig-iron of marketable composition?
3. The experiments made with charcoal as a reducing agent having failed, could the process be so modified that charcoal could be substituted for coke?

These points were of such importance to those Canadian provinces which possessed the ore in abundance, but had no suitable coal for coking, that the Government set aside $15,000 for the purpose of carrying out these investigations.

The book gives an elaborate account of the experiments undertaken, with drawings and photos of plant employed, and complete statements of materials employed and results obtained.

Ultimately the above queries appear to be answered in the affirmative, and at the present time a 3,000 h.p. plant is in course of erection.

What this means to Canada, and to other countries where similar conditions obtain needs no emphasis, and doubtless in the not very distant future will have a very direct bearing on the problem of iron and steel production and utilisation of the Victoria Falls power in this sub-continent, more particularly as titaniferous ores are shown to be utilisable by the process; consequently these experimental results are of great importance.

Probable costs, under Canadian conditions, are fully worked out, and an admirable appendix gives a very useful account of the latest developments and inventions for the electrical preparation of iron and steel, also lavishly illustrated by folding plates, the information given being probably unobtainable elsewhere without much laborious searching. (G. H. S.)

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* See this *Journal*, vol. vii., Aug., 1906, p. 55.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) means provisional specification, and (C) complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)

(P.) 359/07. J. H. Hughes. An improved safety catch for cages, skips, lifts or the like. 21.8.07.

(P.) 360/07. J. Snodgrass. Improvements in the manufacture of cyanides and apparatus therefor. 21.8.07.

(P.) 361/07. J. Snodgrass. Improvements in the manufacture of cyanides and apparatus therefor. 21.8.07.

(P.) 362/07. W. Punt. An improved device applicable to cold lead joints for cast iron pipes. 22.8.07.

(P.) 364/07. J. S. Allison. Improved jockey or clutch and attachment to haulage truck. 23.8.07.

(C.) 365/07. H. Zerning. Improvements in the manufacture of filaments for incandescent electric lamps. 23.8.07.

(C.) 366/07. Ozonair, Ltd. (1), F. Rogerson (2). Improvements in the art of brewing beer and other liquors. 23.8.07.

(P.) 367/07. C. E. Thompson (1), H. H. Johnson (2). Improvements in safety brake gear for mine cages, skips and the like. 24.8.07.

(P.) 368/07. O. F. W. Retzlaff (1), J. V. Lindhorst (2), M. Kindinger (3). Improvements in sash windows. 24.8.07.

(P.) 370/07. A. E. Davis. Improvements in and relating to the dumping of tailings and in the apparatus therefor. 24.8.07.

(P.) 371/07. F. Beard (1), A. T. Harris (2). Improvements in the automatic feed of ores for crushing. 26.8.07.

(P.) 372/07. R. G. Guyon. Improvements in cycle locks. 29.8.07.

(C.) 373/07. K. B. Quinan. Improved method of an apparatus for drying gun cotton, nitro-cellulose and similar bodies. 29.8.07.

(C.) 374/07. J. Richmond. Improvements in spades, shovels, forks and the like. 30.8.07.

(P.) 375/07. W. C. Stephens. Improvements relating to rock drilling machines and to drills therefor. 31.8.07.

(P.) 376/07. J. Jickell. Safety brake for cages or skips when being raised or lowered in shafts. 3.9.07.

(P.) 377/07. A. W. Ferns. Concentrating and alluvial washing machine. 4.9.07.

(P.) 378/07. R. V. Driscoll. Improvements in safety catch mechanism for mine skips and the like. 4.9.07.

(C.) 380/07. P. E. Cumberland. An improved method of preventing the corrosion and decomposition of metals immersed in or in contact with water or other liquids due to electro-chemical action. 5.9.07.

(C.) 381/07. T. W. Capen. Improvements in shaft suspension especially applicable to gyrating crushers. 5.9.07.

(C.) 382/07. G. Westinghouse. Improvements in or relating to yielding resistance devices. 6.9.07.

(P.) 383/07. Aetra Self-Heating Food Co. (1), P. Friedman (2), A. H. Carlisle (3). Improvements in heating cans. 6.9.07.

(C.) 384/07. O. M. Torell. Improvements in rock drilling machines. 6.9.07.

(P.) 385/07. J. Bueb. Improvements in the manufacture and production of cyanides in a condition fit for transport and storing. 6.9.07.

(C.) 386/07. C. D. Burnette. Linings for tubular grinding mills. 6.9.07.

(C.) 387/07. B. Solis. Improvements in and means or apparatus for treating slimes and other substances with air and other fluids. 6.9.07.

(C.) 388/07. D. B. Morison. Improvements in or relating to steam condensing plant. 6.9.07.

(P.) 389/07. J. Sachs. A new and improved chemical preparation for sweetening purposes and mode of manufacturing the same. 6.9.07.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

Banks, E. Gripper, *to* Waihi G. M. Co., Ltd., Auckland, New Zealand.
Beatty, J. W. S., *l/o* Johannesburg; French Rand G. M. Co., Ltd., P. O. Box 30, Luipaardsvlei.
Bell, H. D., *l/o* Cape Colony; York G. M. Syndicate, Ltd., P. O. Box 54, Krugersdorp.
Boer, E. C., *l/o* Sabiwa Mine; Maluti Reef, near Colleen Bawn, Gwanda, Rhodesia.
Bristol, F. A., *l/o* Germiston; P. O. Box 1167, Johannesburg.
Brown, G. D., *l/o* Johannesburg; 23, Queen Street, Horncastle, Lincs.
Christopherson, E. H., *l/o* Germiston: New Goch G. M., Ltd., P. O. Box 540, Johannesburg.
Evans, W. Bowen, *l/o* London; Victorian Cornish Gold Mines, Ltd., Daylesford, Australia.
Hamilton, E. M., *to* c/o Messrs. Charles Butters, Ltd., 818, Merchants' Exchange Building, San Francisco, Cal., U.S.A.
Hamilton, J. L., *l/o* Randfontein; Langlaagte Deep, Ltd., P. O. Box 5, Fordsburg.
Johnston, James, *l/o* London; c/o Messrs. Charles Butters & Co., 115, Broadway, New York, U.S.A.
Krige, W. A., *l/o* Germiston; P. O. Rietfontein Mines, Transvaal.
Lucke, P. K., *l/o* London; 1a, Calle Zaragoza 18, Saltillo, Coah, Mexico.
Marriott, F. A., *l/o* Kalgoorlie; Lancefield G. M. Co., Laverton, W. Australia.
Murdoch, R., *to* Colleen Bawn Mine, Gwanda, Rhodesia.
Normand, J. N., *l/o* Scotland; Surprise G. M. Co., Ltd., Suprise Siding, Gwelo, Rhodesia.
Stokes, Ralph, *to* 34, Priory Road, West Hampstead, London, N.W.
Thomas, John, *l/o* Knights; New West Bonanza G. M. Co., Ltd., P. O. Box 137, Klerksdorp.
Tillard, M. O., *l/o* Langlaagte; Jumpers Deep, Ltd., P. O. Box 1, Cleveland.
Tullis, W. W., *l/o* Rhodesia: Leatherhead Court, Surrey, England.
Wertheim, C. J. M., *l/o* Batavia; Hilversum, Holland.

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OF THE
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OF SOUTH AFRICA.

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Proceedings
AT
Ordinary General Meeting, October 19, 1907.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, October 19th, Prof. J. Yates (President), in the chair. There were also present :—

45 Members: Dr. J. Moir, Messrs. J. Littlejohn, F. F. Alexander, R. G. Bevington, W. R. Dowling, A. Heymann, A. McA. Johnston, A. Richardson, G. O. Smart, Prof. G. H. Stanley, A. Whitby, H A. White, Prof. J. A. Wilkinson, J. R. Williams, S. H. Pearce, S. Beaton, E. H. Croghan, M. Ford, J. C. Greer, Jr., H. R. Grix, W. H. Jollyman, J. A. Jones, M. Knight, G. A. Lawson, R. Lindsay, W. P. O. Macqueen, D. McKerrell, P. T. Morrisby, T. T. Nichol, C. F. Parry, J. F. Pyles, E. T. Rand, C. E. Rusden, S. Shlom, T. W. Simmons, W. H. Smith, J. J. R. Smythe, A. L. Spoor, H. Taylor, W. Taylor, J. P. Ward, H. Warren, F. W. Watson and J. O. Welch.

10 Associates: Messrs. A. Avent, H. G. Brickwell, J. W. Carter, J. Chilton, R. W. Leng, R. K. Maxwell, R. W. Maxwell, C. G. J. Moore, E. M. Weston and L. J. Wilmoth.

10 Visitors and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as published in the *Journal*, were confirmed.

NEW MEMBERS.

Messrs. Croghan and Jollyman were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been duly elected, as follows :—

Angus, William Hay, Knights Deep, Ltd., P. O. Box 143, Germiston. Foreman Amalgamator.

Mance, John Courtenay, P. O. Box 97, Johannesburg. Mining Engineer.

Newson, James Aldrovandi, P. O. Box 113, Klerksdorp. Assayer.

VISIT TO THE JUMPERS G. M. CO.

The President: Since our last meeting we have had a very pleasant visit to the Jumpers mine. I should have liked more of the members to have availed themselves of the opportunity, for certainly those who did go had a very enjoyable and instructive time. I move, gentlemen, that a letter of thanks be sent to the manager for his great kindness.

VISIT TO GOLDFIELDS LABORATORY.

You will have noticed that the metallurgical staff of the Consolidated Goldfields of South Africa have been kind enough to invite the Society to inspect their laboratory. The visit takes place on Saturday afternoon next, and I should like such of you as intend going to be good enough to let Mr. Rowland have your names to-night. I have already had the pleasure of going over the laboratory several times, and I strongly advise any of you who have not seen it to avail yourselves of this opportunity. I am glad to say that Mr. McArthur Johnston is to favour us with a paper to-night on laboratory work.

THE LABORATORY: ITS ECONOMIC VALUE.

By A. McArthur Johnston, M.A., F.C.S.
(Member).

This Society has in the past been the medium of upholding the position of the assayer in gold mining work. Many of us can remember the time when the assayer was looked upon by the management as a necessary evil on whom it was very convenient to rely, if his results tallied with the actual gold won. Happily this has changed, and I think I am in no way overstepping the mark when I say that there is no more valued man on the mine staff at the present time than the assayer, of whom great accuracy and reliability is demanded. Towards this change this Society has done much, and I would therefore crave the indulgence of members in presenting to their notice the work, and the value thereof to the industry, of a well equipped laboratory.

In introducing my subject let me stipulate the scope and duties devolving on this branch. First and foremost, it must be conceived as a sound financial concern. It must be a commercial success, and the ultimate aim should be the testing of all supplies in use on our mines. We all know the state of affairs in Britain, where unfortunately the works chemist is little more than tolerated, but if we go further afield, we find other countries farther advanced in this respect. Germany pays great heed to the work of the chemist, with the result that that nation has, so far outstripped her insular rivals in certain of the arts, whilst the strides made in the United States in all branches of metallurgy is undoubtedly due to the research work instituted by its chemists and metallurgists. The latter country also leads, in that many of the largest companies and railroads buy their supplies on a guaranteed basis, and I learn that a considerable amount of success has followed the adoption of this principle. "Some of the large coal companies," writes *Mines and Minerals*, "such, for instance, as the H. C. Frick Coke Co. already have well equipped laboratories in which not only the coal is analysed, but all supplies such as paints, oils, etc., just as is done now by practically all of the railroads of the country."

Let us consider our case here. We know perfectly well that these gold mines are financial concerns. They buy and they sell. Mark, however, the difference. They buy what is given to them—with restrictions and reservations certainly, but they sell on a definite basis. Their marketable product—the gold—is sold on a very definite footing. Now, on the face of it this seems incongruous. Our near neighbours, the coal mines, are able to supply their wares on the basis of a trial consignment, or on the analysis of a sample of their coal taken across the reef, or it may be on the plausibility of their agents or business managers. The result is that several months' supply of coal is obtained and used before the engineer realises that the anticipated standard is not being obtained, and during this interval considerable friction has been caused with, mayhap, a general clearing out of the firemen. Now, I do not take up the standpoint that the laboratory is and must be the primary factor in determining the class of material to be used. The engineer should come first in this respect, but I most distinctly affirm that the laboratory is and can be of most material advantage if properly used, and it is to point out the various methods which might be adopted to this end that this paper is chiefly concerned.

Unfortunately the laboratory has in the past in this country been looked upon as an expensive luxury. Our financiers naturally expect some tangible return, from an outside source, of the value expended in the initial cost and running expenses, and as it very often happens that such is not forthcoming, its importance is not appreciated. They forget that it is there that most of the initial work in experimenting and in building up data on which to base changes in running operations, is carried out. Many of the results obtained undoubtedly are negative, but if occasionally one or other idea turns up trumps then the industry is the gainer. Looked at in this way, I may safely affirm that these initial experiments cost but a mere trifle as compared with what they would cost were they attempted in actual practice and, as often happens, found wanting.

I may state that the one and chief end of our work in the laboratory is to reach the managers and engineers in such language and facts as they can quickly grasp and utilise. We appreciate that they are busy men and have little time or even inclination to go into all the complexities and details of an ultimate analysis. For example, in dealing with coal samples sent in, we try to show by the proximate analysis, the exact value of the coal in conjunction with the calorific value attainable with complete combustion. These figures give definite standards—comparable with other coals—and we would only complicate matters, not to talk of the extra amount of work and hence, expense, involved, were we to conduct and report an ultimate analysis on every sample of coal sent in.

Having now outlined the basis on which I consider the work of the laboratory should be conducted, I will proceed to give an outline of the various methods in use in the Consolidated Gold Fields laboratory.

As being of more particular interest to the majority of members, I will begin by showing wherein the laboratory may be of advantage to the reduction works. The mill manager, I find, is not above the aid of the metallurgical chemist, and any tests and work done connected with his department are always welcomed by him. Let me instance one case. It is well known that in starting up with new plates there is a considerable deficit in the gold anticipated, the loss being due presumably to absorption by the plates. Now some tests we conducted showed that by rubbing in silver amalgam, the plates can be brought into good condition, and there seems no reason why this should not be adopted in future practice on these fields. A month's preparation of the plates in this manner prior to starting would be well repaid if even only 50 per cent. of the gold usually absorbed were recovered. Practical results may of course not bear out this laboratory test, but the fact remains that we did get good results on a small scale, and with the use of nothing which could not be adopted on a large scale.

The cyanide manager requires his lime tested and his cyanide supply standardised. These two articles the Gold Fields group of mines have for some time now been buying on a guaranteed percentage basis, the former determined by the percentage of caustic lime it contains and the latter by the percentage of cyanide (estimated as KCN) and the absence of sulphides. The sulphides are not supposed to rise above 0·02 per cent. Na_2S, whilst the cyanide should have over 129 per cent. KCN. Little difficulty is now experienced in obtaining cyanide of the requisite strength and purity, but the basis of estimating the caustic lime present in our local supply has been rather a sore point with the seller, though now that the lone hand practised by the Consolidated Gold Fields has been strengthened by the paper read at a recent meeting by Mr. Croghan, we anticipate a more satisfactory basis—satisfactory, I mean, to both buyer and seller. Occasional testing of the working cyanide solution has been carried out, but further analyses in this direction would be advisable, since the time may come when inferior solubility of the gold or incomplete precipitation of it may require research work, and so far as I know the data on working cyanide solutions is meagre enough. Occasionally, too, some change in the ore or the crushing thereof may necessitate a few experiments to find out the best line of treatment, and those of you who know the paucity of time available for experimental work to either the cyanide manager or assayer can well appreciate the laboratory where such tests can be carried out.

A well-known metallurgist on these fields once told me that whenever he went to a new mine the first man he made friends with was the engineer. I heartily endorse this remark, for I can well appreciate that the man who wants, and who can appreciate, the uses of the laboratory is the engineer. His connection therewith should be practically a part of his daily work, and I am sure he will excuse me if I point out to him some phases of the work there which would advantage him and perhaps simplify his work with, I am sure, a considerable saving of worry. Most of these are already familiar to our own engineers, but a perusal may help to renew recollections of the work already done.

At the present time the supply of coal is exercising the attention of all engineers on these fields. A short time ago, as you will remember, an amalgamation took place between many of the colliery owners. The idea was to raise the price of coal sufficiently to place the collieries on a paying basis, since competition had so lowered the price at the pitmouth as to allow a very small margin of profit. The importance of this subject to the industry can be well understood when it is considered that the mines foot an annual bill of £1,175,000 for coal alone. The question then naturally arises—how can the mines act so that they may ensure their getting their money's worth? The first response would be the establishment of definite standards of quality in the supply. Undoubtedly the ideal system would be a guaranteed minimum of steam generated in the boilers per unit of coal used. About a year ago Mr. H. C. Behr read a paper on "Power and Efficiency Tests on the Mine" before the Transvaal Institute of Mechanical Engineers. He outlined there how to test which coal gave the best efficiency for the class of furnace and boiler in use. This information, if rigidly obtained, would undoubtedly be the basis on which the future supply of coal would depend, and it would then be the duty of the laboratory to test this coal that the standard be maintained. Payment by results on steam generated is subject to too many variables and difficulties, the personal factor being very often largely in evidence. These difficulties could undoubtedly be overcome, but the simpler plan suggests itself, viz., a thorough sampling of the coal and the laboratory proximate analysis and calorific value. "Engineers, who have adopted fuel testing," says Kershaw, "as an aid in the work of boiler management rarely relinquish it; and as their knowledge of its usefulness extends, they place more and more reliance upon the laboratory test results, and less upon the steam raising trials, in forming their comparative judgments of different fuels."

Initially then we must get a proper sample. In this there is no particular difficulty, since sampling on many times larger tonnages is in constant use on every mine. One method should be adopted and strictly adhered to. I may say that the sample taken during the discharge into the bins is undoubtedly the most reliable. Engineers must beware of placing much value on colliery sampling, since perhaps even more so than in gold mining do seam samples give an unreliable estimate. Car sampling cannot be always reckoned as absolute, for layers may have formed from different parts of the mine during the filling operations; and again, the lump coal—invariably the best—is liable to remain on the top, the smaller particles sintering to the bottom.

The sample is finally crushed and triplicates taken, two being reserved for further reference if desired. All samples should be placed in stoppered bottles.

In conducting the analysis the evaporative factor is undoubtedly the most important item. This we have always found to give extremely steadfast results when carried out in the Mahler bomb. This method of estimation is undoubtedly the recognised standard and presents little difficulty, whilst eliminating almost entirely the personal factor. I recently made a comparison on bituminous coal between tests conducted by the Mahler bomb and by Rosenhain's improved Thomson coal calorimeter, with the result that I found practically no difference therein.

The proximate analysis calls for the estimation of moisture, volatile matter, fixed carbon and ash. Of these the one calling for most comment is the estimation of volatile matter. By this we mean the volatile hydrocarbons, that is, a part of the carbon and nitrogen, the hydrogen and the oxygen, including the combined water or that not given off at 100° C., and a part of the sulphur. The various methods of estimating the volatile matter are liable to give varying results. Chiefly on account of the proximity of the assay muffle we make use of this and carry out our treatment there. The muffle is carefully dusted all round with bone-ash and the covered porcelain cup containing 1 gm. of the powdered coal is placed on a scorifying dish and put in the muffle. After a minute or so, the time being dependent on the heat of the muffle, the gases generated from the coal escape and burn more or less steadily. Notice is taken when this flame ceases, and within from $\frac{1}{4}$ to $\frac{1}{2}$ minute the crucible is withdrawn, the time in this case also being regulated by the heat of the muffle, a shorter time being required, of course, as the heat is greater. The difference, minus, of course, the hygroscopic water, represents the volatile matter. A further two hours in the muffle, with the cover removed, at a bright red heat is sufficient to get rid of the fixed carbon. Our custom is to place the crucibles for the estimation of the ash in the muffle at the end of the day's assaying and remove them first thing in the morning. In weighing under such conditions it is imperative to tip the ash prior to weighing, since the porcelain crucible is liable to increase in weight from absorption of the lead fumes, or to decrease in weight owing to chipping of the crucible. Since the calorific value of a coal is dependent largely on the amount of ash present therein, I would suggest that coal bought under contract should be sampled daily. This sample could be sent to the assay office during the afternoon and the ash estimated each day, whilst the complete proximate analysis and calorific value could be carried out on a weekly sample made up from the daily samples. A rough daily check could thus be had on the coal supplied, and, as will be seen later, on the ash sent to the dump. It may be noted that in estimating the calorific values a pressure of about 15 atmospheres of oxygen is sufficient for burning our coals and the adoption of 25 atmospheres, the point usually marked on the pressure gauge, is unnecessary. The crushed coal is placed in the receiving vessel without drying, but the calorific value is estimated as the heating effect of 1 lb. of dry coal, this latter being calculated from the proximate analysis figure. In connection with this it may be noted that the oxygen cylinders bought locally, though imported from England, I believe, do not represent pure oxygen. One we tested a few weeks ago gave 2·2 per cent. CO_2 and 86·6 per cent. O_2.

Of great importance, too, to the engineer is the analysis of the ash sample for fixed carbon and evaporative power. As with the coal this sample ought to be thoroughly representative. There are, however, several considerations to be taken into account when basing conclusions on the figures obtained. First and foremost, one must reckon on the quality of the coal used. It is pretty generally acknowledged that a coal having an ash content of over 40 per cent. is not worth burning, and the nearer the coal used approaches this, the worse will be the ash discharged. There ought, however, to be little difficulty in obtaining an ash having a carbon content of under 30 per cent. when working with one of the better class of coals available here, viz., with about 15 per cent. of ash. The evaporative factor of the ash can be estimated in the Mahler bomb in the same way as that of the coal, but I find it advisable to use a higher pressure and to carefully spread out the sample in the platinum receptacle. The formula we use for estimating the percentage of extraction of the combustible matter is :—

$$100 - \frac{\text{E.F. of dry ashes} \times \% \text{ of ash in dry coal} \times 100}{\text{E.F. of dry coal} \times \% \text{ of ash in dry ashes.}}$$

On these figures an extraction of over 90 per cent. should be easily obtained. We have tried to get for these fields a formula on which to base our calorific value by calculation from the proximate analysis, but so far have failed. In several cases we can approximate it very closely, but practical results have over and over again foiled us, so that we place little reliance now on such calculated values. Indiscriminate mixing of the coal from various parts of the mine is undoubtedly the cause of failure.

Passing from the consideration of coal, the next most important subject, generally speaking, is the feed water supply for boilers. Some recent researches on the loss of heat due to boiler scale describe a loss up to 10 or 12 per cent. from a scale thickness of $\frac{1}{8}$ in. It has also been concluded that the mechanical structure of the scale has quite as much, if not more, effect than the chemical composition. The engineers who get the use of town water are lucky. To the majority this is denied, and they have to make the best of the mine water obtainable either from their own or a neighbouring property. In very few cases is this water equal to boiler feed requirements, and I have little hesitation in affirming that a water purifier is an absolute essential on all producing properties here. Not a few have been erected but, through misuse or neglect, they have failed to give satisfaction and are now in many cases derelicts. The services of the chemist are undoubtedly of advantage here, and less attention should be given to the rule of thumb man. Summarising my experiences on these fields, I can safely affirm that the chief difficulty is to obtain a thorough mechanical mixing, either by injecting steam or stirring, between the soluble solids in the water and the chemical reagent added, so as to bring about the desired reaction.

I have found that the invariable result is the entrance of the alkali into the boiler, and this in time brings about a state of affairs worse than the first. It is almost useless to depend on the self-feeding mechanical arrangements which have been so generally adopted. Theoretically, they give splendid results, but rarely do they get the necessary attention that the highest purification can be guaranteed. What we require is a water softener, which will by heating or agitation bring about an intimate mixing of the reagents with the solids in solution, and a careful superintendence thereof. The water in the mine changes month by month, and there should be therefore a regular analysis of this and of the purified water if the best results are to be obtained. For comparative work, a rough and ready guide to the engineer who cannot avail himself of the more satisfactory complete analysis, is Clark's hardness test with the soap solution, though a little practice is necessary for carrying this out. The degree of hardness to be aimed at should be well under 4. The deleterious effect of oil on boiler plates is so well known that this should always be looked for in feed waters.

With regard to the addition of boiler fluids to the boiler I have little to say. They are but a makeshift at best and their use is soon discontinued.

Analyses of boiler scale are also of service to the engineer, and in estimating this it must be remembered that our mine waters contain gold in solution. Hence boiler scale may and does generally contain a considerable amount of gold and silver. The result is, however, more interesting than payable—at least let us hope so, for our engineers' peace of mind.

Let us next consider the question of lubricating oils. The most important points to be considered in the testing of these are the flash point, the percentage of fatty oils present, the acidity and the gumming. A high flash point is a desideratum which all engineers strive after, and especially with high class cylinder oils. For the latter there should be no difficulty in obtaining a compressor oil with a flash point of over 500° F., and this will undoubtedly be a necessary standard in the near future on these fields, since it is only by adopting the best class of oil that fouling of the compressor air can be avoided. The question of the percentage of fatty oils present and of acidity calls for little comment since all good oils seldom fail in this respect. The gumming test is, however, rather important, as we have found that oils failing in this test gave very unsatisfactory results in actual work. The film formed by some oils, and which prevents easy working of the lubricating surfaces, is due to the presence of rosin oil, to oxidation, or it may be to the presence of soaps formed by the chemical action of the free fatty acids on the metal of the bearing, dissolving in and thickening the oil. It may be noted that it is only by fractionating the oil that the presence of resins can in many of the lubricating oils be detected.

Although the lubricating power of an oil is closely related to the viscosity, we place little importance on the estimation of this, for all our engineers are good at estimating for their respective requirements the necessary viscidity. A set of figures giving the rate of flow of 50 c.c. through a Boverton Redwood viscometer conveys but little to the working engineer, though I may mention that the figures we do give are reckoned on the rate of flow at 60° F., 75° F., 100° F., and 150° F.

Rope greases also call for some attention. In this case the desired information would include primarily the presence or absence of grit or body likely to set up irritation or to interact chemically with the bearing surfaces. Acidity must also always be looked for, and more especially when there is present a disguising cloak in the shape of nitro-benzenes.

Of considerable use also to the engineer is the Orsat apparatus usually found in all laboratories. Many of the most up-to-date of our mines are equipped with self-registering carbonic acid apparatus, and the Orsat is periodically of use, not only in checking these but also in determining the carbon monoxide percentage. This latter engineers are occasionally inclined to overlook, forgetting that a high percentage of CO_2 is not always consistent with good work, since not infrequently CO may be present, and, of course, is not detected by the self-registering instrument.

The more accurate Hempel's pipettes are also necessary appliances, our greatest use for them so far being in the estimation of mine gases.

Apparatus for the registering of heat generated in furnaces is a decided gain to any laboratory. A set of Saeger cones we stock for reference and check work, but the most important, because handy, instrument we have for this purpose is the Féry pyrometer. The reading can be taken at a convenient distance from the furnace, since the appliance works by radiation, and there is little danger of destroying the instrument, provided ordinary care be taken.

The testing of candles is a most important one when we consider the enormous quantity used, the figures for the last twelve months, as shown by the Government Mining Engineer's report, pointing to a consumption in gold mines of £208,000 worth. In conducting this examination practical tests are conducted, the aim being to determine their value in actual working conditions as approximately as possible. The intensity of the light, the height of the flame and the melting point are, of course, undertaken, but as none of the candles so far tested by us have failed to pass the standard requirements, we put more importance on the actual burning, viz., the cost per hour consumed in still air and the price per hour consumed in a draught of about 60 ft. per minute, noting the guttering of the wick and the hardness or softness of the candle. Comparisons are made with the standard English sperm candle, and the draught is measured with an anemometer placed alongside the burning candles.

Primary importance also should be given to the testing of Portland cements. The quantity consumed in this country last year, amounting to £107,000 for our gold mines alone, is a sufficient excuse for giving this subject our earnest consideration. There is no doubt but that there have been imported, and still is being imported into the Transvaal, cements which have been shunted here, simply because they have failed to pass the requisite standard specifications in the land of their birth, and knowing that they are seldom tested here, they have been shipped with, let us say, the pious hope that the sea voyage would benefit them considerably. The engineer has experienced this to his sorrow, when he finds that to begin with they fail to unite with the accustomed proportion of rock and sand, and later on when he sees his foundations going. It does indeed seem strange that cement, the foundation work of much valuable plant and machinery, should have received so little attention, especially as the testing of this is so vigorously adhered to in European countries and the United States.

The tensile strength is undoubtedly a good test on all cements, but there is nothing in this test which good cements have any difficulty in negotiating, indeed, my experience has been that the best imported cements seem to have been built to pass these tests. On the other hand, the poorer cements fail badly, the most common cases of failure being apparent in the sand test. I distinctly favour the adoption of the English standard specifications, since the hand ramming with a competent workman is not so liable to be overdone as mechanical ramming. With mechanical ramming, too, about one-fifth more material can be pressed into the mould, and this should call for a higher tensile strength. It may be urged that ramming is adopted in actual practice, but as against this we must remember that the necessary qualifications for a good cement are comparative, not with actual conditions, but with other laboratory results. If mechanical ramming be decided on, then it should be imperative that a machine giving the same number of blows and exerting the same pressure on all samples be used. The sand we employ in making these briquettes is clean quartzite passing a 20 mesh sieve and remaining on a 30 mesh and thereafter washed and dried prior to use. This gives a high percentage of voids, but the results obtained in using this sand are very satisfactory, though, of course, only comparative with other work done here.

There seems to be some dissatisfaction as to the value of the soundness test. The best test, of course, is time, but this is not always practicable, and we must resort to heat to give us the desired information. Pats of all descriptions are made, but sufficient notice is given to the subject when we mention the Le Chatelier tongs test, as recommended by the British standard specifications. This calls for a six hours' boiling of the cement, after 24 hours under water, and a consequent maximum of expansion of 12 mm. So far as we can learn there have been manufactured cements which have failed to pass this test and yet have withstood the lapse of time, but as against this we find the British Committee this year recommending the reduction of even the present maximum of expansion. The action is not well understood, but there is safety in the advice of Taylor, who concludes that it is well never to use a cement failing in the accelerated test, when one equally suitable can be obtained passing this test. German manufacturers and chemists, who have done much good pioneer work with cement and the testing thereof, favour heat tests. The specific gravity test, the fineness and the chemical tests are all useful, but there are few Portland cements introduced into this country which fail to pass the called for requirements. Most of the cements met here are slow setting. A demand has set in for some quick setting cement, especially for tube mill setting of the blocks, but I do not know of any here to satisfy that class of work.

Of considerable importance, too, to engineers is the quality of the rubber imported into this country and sold by the agents as pure. Recently we tested a lot of presumably best rubber which contained but some 10 per cent. of rubber. The filler or mineral matter of this material was composed of barytes, chosen perhaps on account of its high specific gravity and because rubber is sold or, is it better to say, bought by the pound avoirdupois. Also, some rubbers may contain as their filler a metallic compound which would be utterly unsuited for the work for which they are intended. For example, if zinc oxide is the inorganic ingredient in a rubber exposed to water containing sulphuric acid, the rubber deteriorates quickly, whereas the destructive action of all oils upon india-rubber is more particularly checked by the presence of zinc oxide.

Another most important item in mining economics is the supply of explosives on which well over £1,000,000 is annually spent. On account of the hardness of the rock composing the matrix of the gold, practically only the most powerful explosives are used on the Rand. The stringency of the laws governing the manufacture and importation of this commodity makes the testing of the same here of but secondary importance. The best blasting gelatine obtainable is at hand for the mines and failure of it is infrequent. The most common test carried out at the laboratory is the ascertaining of the relative blasting powers of different explosives. These we obtain with the aid of the lead cylinder or crusher gauge and the foot pounds machine diagram introduced by Lieutenant Quinan. In passing, I may say that we fitted up an apparatus for checking Quinan's curve, and our results, which we carried to a higher degree of compression, thereby elongating the curve, agreed very closely with his, any little differences apparent being, no doubt, due to our carrying out the tests in the open air. For the analysis of explosives the laboratory is also equipped, but the non-traceability of accidents to inferior or faulty explosives has rendered the work done in this direction so far of little service. It will have been noted, however, of late that mercury bichloride has been added to some high-class explosives with the ostensible purpose of preventing moulding, but incidentally masking the heat test results. Since great reliance is placed on this heat test, the presence of mercury should always be looked for.

Of more value is the testing of safety fuse. According to the figures of the Government Mining Engineer safety fuse, valued at £118,000, was imported into this country for use in gold mines last statistical year. Fortunately for the industry the testing of this commodity is carefully superintended by Government inspectors, and only once have I had to condemn fuse which had recently been imported and supplied to our mines. We happened to strike a bad case. Apart from this we have had to report adversely on safety fuse which had been stored for some little time in this country. Chemical members will know from experience how quickly rubber goods perish here, and the waterproof material used in safety fuse very speedily deteriorates, with the consequent result that it fails to pass the pressure test under water. Only one fuse has, so far as my experience goes, stood the test of lying stored for a long period in this dry climate and altitude. Others of the same age have failed, some of them miserably. The value of this testing is appreciated by our underground managers, though there are some who claim that the best way to test fuse is in actual experience. I grant this entirely, just as the best way to get the true value of a gold mine is by crushing, amalgamating and cyaniding the ore. The laboratory method is quicker and less likely to lead to bad results in breaking ground, and though, of course, only a small part is tested, more attention is given to the carrying out of the test than is possible underground, thus rendering the result more conclusive and not exposing the fuse to undue prejudicial handling.

Recently we have introduced into our laboratory apparatus for the microscopic analysis of metals. On this subject I intend to say little at the present time, as on some future occasion I hope to bring before you details of some of the work already done on the microstructure of metals and alloys. As most of you are aware the examination of metals is conducted under three headings, chemical, mechanical, and structural. The latter science is but in its infancy, but its growth is extending rapidly, and the numbers of eager workers, who yearly contribute the results of their researches, augurs well for the future utility of the science. Engineers and mill managers in this country have often had some of their best work rendered nugatory by the breaking of some difficultly replaceable part of some machine, and an indignant—speaking mildly—reference to the agent, and through him to the maker leads to nothing satisfactory, if we except the satisfaction that the manufacturer has in writing that the man out here is not competent to judge whether the breakage is due to bad material or not. Microscopic analysis aims at getting at the root of this evil, and in combination with the chemical analysis and mechanical tests there is no reason why the mines should not get some of their own back. In this connection I may say that so far we have been able to condemn, and demonstrate the cause of failure of, not a few of the castings and steel imported for use on our mines. Let me instance one case at the present time, and in doing so show you by means of the accompanying photo-micrographs what I mean. Two sets of shafting were supplied to one of our mines by different makers. The one lot has done and is doing extremely good work, whilst the other set broke after being in use for some days. Both were sold as low carbon steel, and the chemical analysis showed that both contained practically the same amount of carbon. A microscopic examination, however, revealed the cause of the trouble. The sample was polished on jeweller's rouge and examined under the microscope. It was then photographed under a magnification of about 70 diameters, and I have here for your inspection the results.

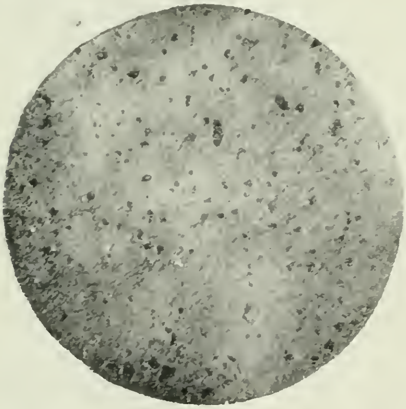

Fig. A.—V.

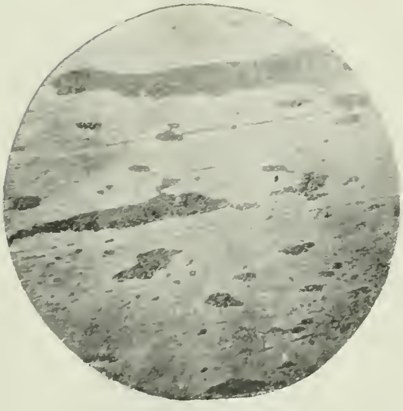

Fig. B.—× 70 Diameters.

The first, Fig. A, represents the good steel, and the surface is seen to consist of the metal with very small particles of slag, all more or less of a rounded shape and evenly distributed, whereas the surface of the bad material, Fig. B—in reality wrought iron—is broken up by numerous lakes of slag. This constitutes the cause of weakness, and as it thus proved that wrought iron has been supplied instead of steel, we are quite justified in demanding a replacement of the shafting. Other work connected with this subject I hope to place before you later, but this example may tend to illustrate the utility of photo-micrography in the detection of inferior steels and iron supplied.

Another branch to which we have devoted a considerable amount of attention in the past has been the treatment of ores from the outside districts. This has been attended with a very appreciable amount of success, since I am pleased to say that our recommendations have been adopted and have borne good fruit.

Again, one of the most important of the many questions to be decided in the laboratory is that of keeping our metallurgical ideas up to date. All of you know how often, on reading of some new development on these or other fields, the desire to test this on your own account, or to prove its applicability to some of your own problems, has assailed you. Necessary apparatus for conducting these trials on a small scale is usually wanting on the mine, and a request for trying them under working conditions generally results in a curt demand to keep down costs. Set-backs in this direction are certainly not conducive to individual effort, and the heads of our industry recognise that more concerted action in this direction would be advantageous, but hitherto the sum total of this recognition has been its recognition. An "experiments fund" was inaugurated some two or three years ago by our largest group here and, quoting a local weekly, it seems that already £15,000 has been expended in trying new devices, and that in one case an unsuitable device cost £4,500 prior to its being abandoned.

Now, I have no hesitation in saying that quite a considerable portion of this could have been saved had the initial tests been carried out on a laboratory scale, meaning thereby a mechanical as well as metallurgical laboratory. I may be permitted to draw the attention of members, however, to one source of false conclusions which are only too liable to crop up in initial experimental work, that is, the test must represent the absolute duplication of the working conditions. That many failures on the large scale are undoubtedly due to neglecting this is only too true, and an impartial test by a thorough practical man should always be insisted on prior to money being spent in large tests. We know how prone the inventor is to overlook small details, which in themselves may latterly cause the failure of the trial. From the point of view also of the originator we find that his grievance is, that on a mine sufficient attention is not given to the carrying out of his ideas to the best advantage. As against this, the engineer, mill manager and cyanide manager usually have sufficient daily worries of their own and are sometimes inclined to afford scant courtesy to the man who increases these worries, more especially if he be a non-practical worker.

A central laboratory would, I am sure, eventually save the industry some thousands of pounds annually, and this Society ought to take a leading part in its initiation. Personally, I am inclined to think that most of this work should be carried out under State supervision—financially I mean—for undoubtedly the solution of some of our problems would lead to the enhanced life of our mines and the opening up of reefs at present unpayable. In this connection I may point to the good work being done by our great rival in gold production, the United States of America. "There can be no doubt that," says a weekly journal devoted to mining, "the United States Geological Survey Department is doing an enormous amount of really excellent work, and if it costs money the authorities have at least the satisfaction of knowing that the value of their liberal policy is fully appreciated not only in America, but also by technical readers throughout the world." When will such a paragraph be written concerning the gold mining problems of the Transvaal?

Gentlemen, you may think this a long introduction to laboratory work, but I mean it only as an introduction. There are many more things I should like to lay before you in this connection, and I hope some of you would like to ask for further information. My object in writing this paper is to bring the laboratory before your notice, and if any of you feel that you would desire a more intimate knowledge of the work done there, I shall be pleased to see you, and by actual experiments demonstrate the value of laboratory test.

Before closing I should like to record my thanks to one of your Past-Presidents, Mr. W. A. Caldecott, for permission to write these notes and extend this invitation to you. He has been indefatigable in having the laboratory fitted up with the best and latest mechanical and chemical appliances, and the success which has attended it since its inauguration is due entirely not only to to the keen interest he evolves in all the work there, but to his personal magnetism in keeping us all going.

The President: We have to thank Mr. McArthur Johnston and the Consolidated Gold Fields Company for a paper of exceptional interest, which I am sure you have all appreciated just as much as myself, and that is saying a great deal. He has, I think you will admit, made out an ample case for the testing of all our materials and stores, and in view of the great sums involved I fail to see how it is possible to adopt any other course than to test everything thoroughly. I do not feel called upon to say more concerning this matter to-night, for I feel sure that we will have an interesting discussion.

Before sitting down I would point out that Mr. J. R. Williams is with us again to-night. It is a real pleasure to see him back again and looking so well and hearty.

Mr. J. R. Williams: I beg to thank you, Mr. President, for your kind remarks, and I can only say that I am sure I am equally delighted to be back again in the Transvaal.

NOTES ON SMALL STOPE DRILLS.

By E. M. Weston (Associate).

Facts must always be of more interest to this Society than opinions. I have, I am afraid, few new experimental facts to bring forward, and this paper must be considered merely as an attempt to collect and bring up to date our knowledge of this stoping problem, and to furnish material for discussion. One cannot help being amused at the fact that it is taking the efforts of a painstaking and very inquisitive Government Commission to rediscover facts about the Mining Industry, and especially about the conditions of rock drill work here, that were fully discussed by this Society in connection with a former paper of mine, read two years ago. The Commission will certainly not suffer for want of opinions, even if, among other things, they are of a radically contradictory nature regarding small drills. In a former paper I advocated the use of higher air pressures, and expressed my belief in the capabilities of small drills from $2\frac{3}{4}$ in. down. I also contended that three $2\frac{3}{4}$ in. drills worked with the same air pressures would do more work than two $3\frac{1}{4}$ in. drills, even in stoping in large stopes. This was generally denied. I believe the experiment was tried by Mr. Ford on the Meyer and Charlton mine, and the final result was that it was found better and cheaper to run with two $2\frac{3}{4}$ in. machines than with two $3\frac{1}{4}$ in. machines. Six holes are now drilled, per shift, with each machine. This experiment throws a great light on much of the evidence that has been given before the Commission. It shows that where in any mine the men are given three machines and told to work them, regardless of the fact that thereby the air pressure is lowered 10 or 15 lb. per sq. in., the efficiency doubtless does fall off. Practical miners who have given evidence before the Commission that they could get as much work out of three machines as out of two were, no doubt, in many cases, right: but such evidence had really nothing to do with the real question, which was, could a white miner supervise three machines efficiently in certain cases? Two years ago Mr. Polglase pointed out that increase of fathomages broken did not correspond in the same ratio with increase of drills used per man: but he did not, I think, point out the real reason. This is not, I think, due to the fact that the supervision is necessarily poorer, but to the progressive lowering of air pressures. This trouble has been partly due to the big mill policy. A manager had a battery of 100 stamps to feed, and he could break enough rock with the machines at work and yet keep his air at some reasonable pressure. It was decided to put up another 20 or 50 stamps—to reduce costs, of course. No doubt, theoretically, the mine's compressor was supposed to be able to run enough extra drills: but leakage losses may have been anything from 10 to 60 per cent. The manager had to put extra drills to work. He gave his men three, who had before been working two, and had to put a few extra machines on a pipe line already too small for those working on it. The drilling pressures dropped far below efficiency point, and directors wondered why the extra stamps did not reduce their costs to the extent expected. It is thus evident that in many cases the working of three machines per man may show a loss, and in some mines it would doubtless pay to stop 10 per cent. of the machines at present at work, or to replace them with smaller sized ones, or ones having a lower air consumption. To give engine room pressures as working pressures underground on the Rand, and to estimate the compressor power on the Rand, without allowing for leakage losses, is not considering all the facts.

Practical miners naturally take advantage of such conditions to tell the Commission that they cannot efficiently supervise three machines. This, I know from actual experience, can be done where the conditions are favourable, and where six holes, if necessary, can be drilled from one rigging up, or where only four or five holes are put in per machine. All this apparently irrevelant discussion has a very direct bearing on the small drill question; because if small drills of any type are to be a success, and to compete with hand labour, or replace it, one white man must supervise three or more; or by using the highest practicable air pressures, the rate of boring must be so increased that it will pay to employ one white man alone with each machine. In most mines, to try and displace hand labour with small machines, with the present compressor power and pipe lines, would spell disaster. Very conflicting evidence has been given before the Commission regarding the value of small drills. Generally, they have been condemned on past performances, though good results have been claimed in certain cases. The piston drill has done good work on the Crown Deep, the Geldenhuis Estate, and the Jumpers mine, and others, and the Gordon drill has had success at Randfontein, Kleinfontein, and other mines. It is urged against the present types of small piston machines, that maintenance costs are as high, or higher, than large machines, owing to the endeavour to reduce weight at the cost of strength; that operating costs are very little different from those of large machines, as the miner in the past grumbled at being asked to look after more than two of them; that power costs came out the same as with large drills, because in small stopes a greater footage of boring must be done to break the same tonnage of rock as would be required in large stopes. Costs are stated to be as high, or higher, than with hand labour on the average, though it is conceded, that in soft ground at the Jumpers, the Geldenhuis Estate No. 3 level, the Nourse Mines, etc., they may show an apparent economy.

To me, there seems to have been various obvious reasons for the failure of small drills in the past in many mines. Apart from the hindrances to good work, touched upon in a former paper, the chief reason was that the running of small drills in narrow stopes calls for more skill, and what is perhaps more important, harder work, more running about, and closer supervision than running two large drills in large stopes. Hence, under the conditions prevailing before the strike, it was considered pretty well a disgrace to run small machines. It was scarcely likely good miners would bother with them, when they could earn more money sitting down comfortably watching two machines at work in a large stope. They preferred often to throw the small machines down the stope. The machines were thus very soon handed over to the tender mercies of the inefficient and the loafer. No one was told off to study their working and effect needful improvements in their operation. Then again, many of the machines were poorly designed and made of poor material.

Messrs. Holman are now making an improved small drill of totally different design, so they will perhaps forgive me for saying that their old $2\frac{1}{2}$ in. diameter tappet machine, was about as bad a machine as could be made. I worked three of them for a year, and perhaps an account of some of their ways of breaking down may be of value, so I give it. First, the machine had a taper chuck, in which the drill tightened itself and had to be knocked out with a drift. The drill bits used in them had to be tempered hard on the corresponding tapered shank. This made them brittle, and they were always breaking in or near the chuck. If they were not sufficiently hardened, they burred up or twisted on the end and could not be taken out. Sometimes the chucks themselves split, owing to poor metal. The worst was, however, that the bits were always sticking tight in the chuck. It was quite a common thing to have one stick fast just as one was hurriedly trying to finish a hole at the end of a shift. It was impossible to use them at all without liners of tin, which the rock drill fitter made for us, or which we cut out of old oil tins; but even with these I have spent hours trying to extract a drill. I have lit fires of candle ends and candle boxes in the stope to heat the chuck, and have, even in desperation, blown out a drill end with a detonator and a shaving of gelatine. Both springs and ratchet were of bad steel and constantly broke. The cradles broke at the neck of the seat. It was almost impossible to keep the valve chest packed tight, and one often had to stop and repack it with a time card. When U bolt chucks were obtained, the U bolt was too weak and was always breaking, and the pads, or keys, were too soft, and always wearing out. With all these drawbacks, and only moderate air pressures, these machines could bore well. They broke half a fathom each, per shift, in average Main Reef Leader ground at 1,000 ft. deep. All these defects could have been easily remedied, and I consider it nonsense to say that a machine for work in 38 in. stopes cannot be built to stand up to the work, even in the hardest rock. My quarrel with the manufacturers of small piston drills is that they do not seem to know what they want, or what the mines want. Are the machines they sell to-day, one man or two men machines? If one man machines, why not arrange them so that one man can run them? If two men machines, what is the sense of cutting down a pound's weight here, and another there, and weakening the machine to breaking point? Surely, they must know that a machine weighing 140 lb., is practically just as easy for two natives to handle as machines of 90 and 100 lb., and can bore quicker, and cost less for repairs.

Until very recently, the 2¼ in. small drill was the standard small stope drill all over America and the United States, and is still used in development largely in U.S.A., Mexico, and Canada. The drill is worked by one man, with perhaps occasional assistance in rigging up, and, theoretically, there is no reason why a small drill cannot be employed as economically as a large one; for if the parts are smaller, the stresses met with are also reduced.

Two systems of work can be employed in narrow stopes. It has generally been denied that holes over 5 ft. long can be economically employed in small stopes of under 42 in. Indeed, some appear to believe that 48 in. is the limiting width for stoping economically with machines. This I know to be a mistake, as we employed holes up to 7 ft. long, and yet kept a stoping width of 38 or 40 in. This method involves the use of 2½ to 2¾ in. machines, and two natives to a machine, with one white man supervising three machines. Four would be too many in ordinary situations. In stopes where the reef itself is 30 or 36 in. wide, I think this method of stoping, with good air pressures and working the stope as previously described, can be made the most economical known, especially if a 2¾ in. Konomax drill with decreased air consumption was found suitable and could be employed. The machine used can easily be made as solid and heavy as necessary, and the repair bill be kept down. One white man can supervise three machines, even with a shifting labour supply, if he takes the trouble to teach his natives, and to explain what is required of them, and why it is required. I always found the Kafir as intelligent as the Chinese, if pains were taken to instruct him. When two or three leading natives left, I always had spanner boys competent to take their place, and thus always had skilled boys under me. The steel used in average ground would be, starter, 18 in. long, star bit, 2 in. diameter Second, 36 in. long, star bit, 1¾ in. diameter. Third, chisel or star bit, 1⅛ in. of 1¼ in. steel, 1½ in. bit 4 ft. 6 in. long. Fourth, chisel, 1 in. steel, 6 ft. long, 1¼ in. bit. Fifth, chisel ⅞ in. steel, 1 in. bit, 7 ft. 6 in. long. In hard ground, sizes of bits would be ⅛ in. or ¼ in. larger all round, and the final steel would be 1 in. in diameter with an 1⅛ in. bit. To employ such long steel requires greater skill, and even then, stope widths in certain ground would tend to to be too high, but should in most cases be easily kept below 42 in., which recently was given as the hand stope widths on the Main Reef Leader in a group of mines. In heavy ground, requiring timbering, it is, of course, an impossible method. It is sometimes forgotten that lashing should be 6d. a ton cheaper in a stope of 40 in. than in one of 30 in., and that the excess widths of machine stoping are generally caused by large blocks falling out of foot or hanging wall, and these cost very little to stack, or roll down out of the way. The other system calls for the use of the one man machine proper, and such machines must be made as light as possible. They may in the future be used in two ways. It may be found economical to run with one white man, gaining increased efficiency and lesser maintenance costs, to make up for higher labour charges. The number of feet that such a machine must bore per shift, to allow of paying for the labour of a white miner, and to make costs compare with those of hand stoping, can easily be worked out for each individual case. In considering whether this can be done, we must remember that records made in the shallower levels of the Geldenhuis Estate and Jumpers mines are no criterion of what can be accomplished in the harder reef of the deeper levels. This also, of course, applies to hammer drill performances at Randfontein and Kleinfontein.

One man drills, run either by natives with four or more in charge of one white, or perhaps, with one white per machine, and boring short holes in stopes of minimum width, are undoubtedly needed to break much of the thin, rich reef on the Rand. With the method mentioned of employing long holes, one rigging up per shift, will generally suffice, as five or six 6 ft. to 7 ft. holes can be bored from one bar, if the stope is properly benched and the bar properly rigged up, as previously described. The problem of taking down and rigging up again, becomes important with one man drills. A miner who is going to bore ten 3 ft. or 4 ft. holes must rig up 3 or 4 times. Even with a 3 in. bar and a clamp and arm arranged as in the Gordon drill, this takes time. A miner, in charge of such machines, should be provided with a selection of spare bars of different lengths, and can then set up fresh bars, ready for his workers to move on to, while his machines are at work. A change from one bench to another can then be made with only ten minutes' delay. This matter is often neglected with big machines. The battle of types between the various small drills is only beginning, and there are yet no definite results to base comparisons on. The question of the ultimate supremacy of the piston or hammer type for work on these fields remains to be solved. Undoubtedly the feature of mining work in America has been the rise and development of the small pneumatic hammer drill. It has undoubtedly displaced the small $2\frac{1}{4}$ in. piston machine to a very great extent in stoping work there and in shaft sinking. In England record footages in sinking in limestone have already been recorded by their use.

Hammer drills may roughly be divided first into two main classes. Those which are valveless, with the differential piston or hammer itself acting as the valve. Machines of this class are the Little Imp, the Hardsoeg Wonder, the Murphy, the Sinclair, and the Leyner Rock Terrier drills (Fig. I.).

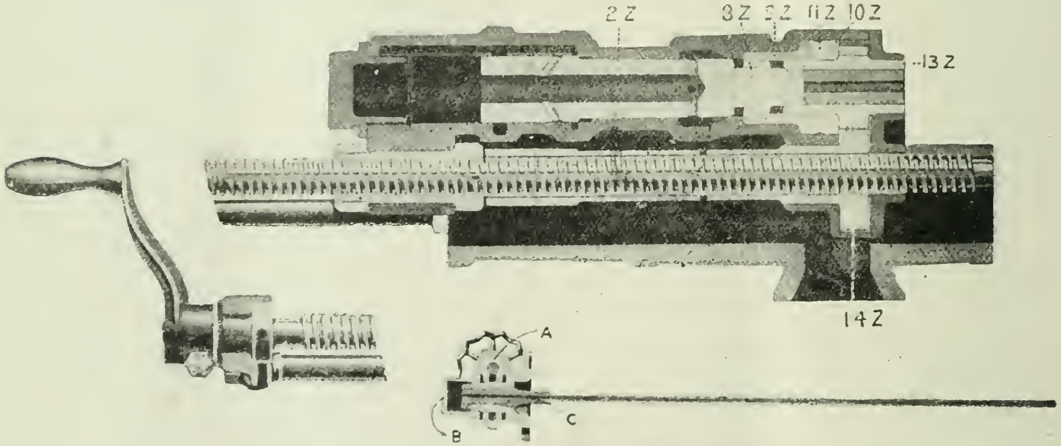

FIG. I.—LEYNER ROCK TERRIER DRILL.

No. 13z is the chuck, formed to receive drill steel of the ordinary cross type.

No. 11z is a buffer, 10z a buffer plate, and 9z the bearing for chuck and for hammer pin. These three parts together serve as a cushion for the striking pin and hammer upon the forward stroke. On its backward stroke the piston cushions upon air.

No. 2z is the piston and 3z the striking pin. Through the latter the blow of the piston is transmitted to the steel.

The outer part of the chuck (13z) forms a pinion, which meshes with the rotating gear (14z). Within this rotating gear are two feathers, corresponding to the two opposite longitudinal grooves in the feed screw.

The turning of the crank, operating on the feed screw, and thus, through the rotating gear, upon the pinion teeth of the chuck, effects the rotation of the chuck and steel.

WATER FITTINGS UPON THE DRILL.—Entering the water connection (A), the water passes into the water connection cap (B) and thence into the water tube (C), which conveys it, without contact with the working parts of the machine, through the drill into the shank of the hollow drill steel. The water tube is much smaller than the passage through the steel. Around the tube, through the same passage, mingled with the water, compressed air is discharged through the steel. A simple needle valve (hand wheel of which is shown in cut) controls the discharge of water.

The large Leyner machine is worked by a piston valve (Fig. II.), and the Little Jap by an axial valve, and the Gordon drill by a piston or spool valve at right angles to piston. They may again be divided into those drills that employ an anvil block to take up and transmit the blow of the hammer, as in both Leyner machines and the Gordon drill, and nearly all the rest which strike the blow directly on the steel itself. They might also be divided into short and long stroke machines, but I will return to this feature later.

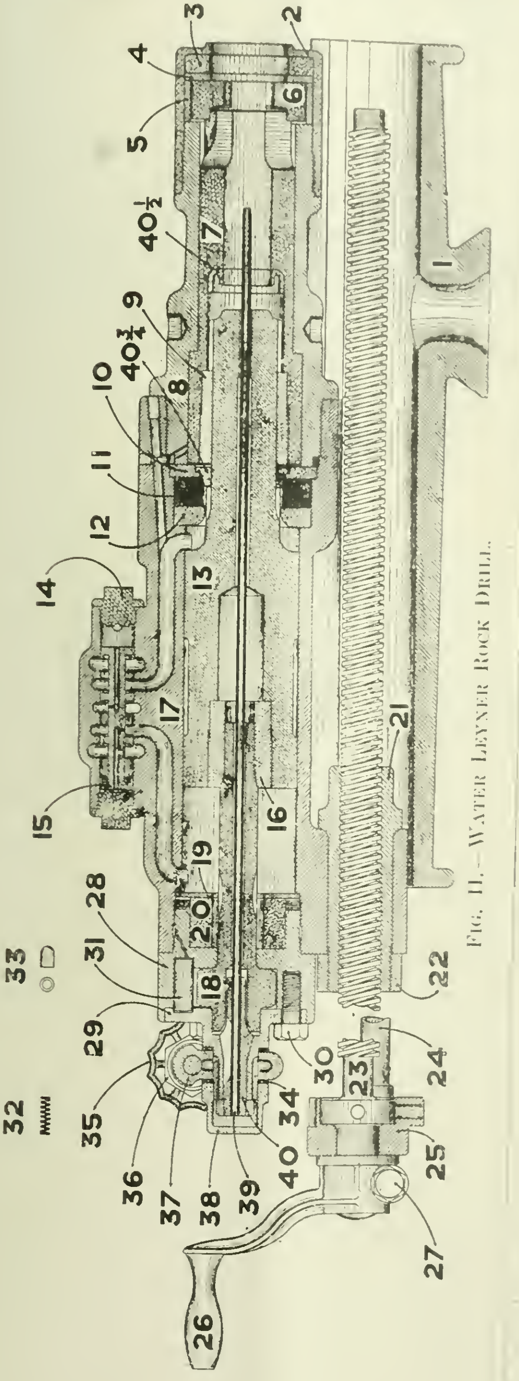

FIG. II.—WATER LEYNER ROCK DRILL.

The favourite small machines in America are certainly those without a separate valve, and which strike the steel direct. Their advantage consists in their extreme simplicity and lightness. They transmit the force of this blow more directly to the rock, having to overcome the inertia of the drill steel only; but they work generally on a short stroke and rely on a high velocity to give momentum to the hammer, which is light, as otherwise trouble might occur through the burring up of the shank of the drill bit, and, as they are often worked held in the miner's hand only, heavy blows would cause too much jar on the operator's muscles. The pistons can be expected to last a long time without undue leakage, as they are ground into a case hardened cylinder; but with grit getting in, leakage losses may be developed quickly.

In the valve machines of small size trouble has been caused, and doubtless will be caused, despite the use of strainers, by small pieces of grit or scale introduced from air pipes or by careless handling getting in and choking the narrow parts or sticking the valve. Some mines in America had to throw out the Little Jap machine on this account. Valve moved, anvil block machines are heavier and and more complicated, but they have a better valve motion, and air can, if necessary, be used expansively. They do not burr up the end of the steel, but they require anvil blocks of special quality steel or are troubled by constant breakages. Differences are also found in the manner in which rotation is affected. The large Leyner machine uses a ratchet and rifle bar like those of piston drills. The Leyner Rock Terrier drill uses a positive geared rotation worked off the feed screw. The Gordon machine has a positive rotation worked by a central spindle ending in a handle and ratchet at the rear of the machine. Most of the other drills are pivoted on the handle or air feed, and are oscillated or revolved by a handle on the front which often contains the air valve. The drawings show a typical make of a simple American hammer drill and its air feed attachment (see Fig. III.). Nearly all these makes of drills are working with success in America, and most of them have been tried here without success. The big Leyner drill is at present, I believe, at work on only two mines, the Jupiter and the Consolidated Main Reef. The Rock Terrier drill is not at work anywhere here. What are the causes of failure? Why is a machine that is

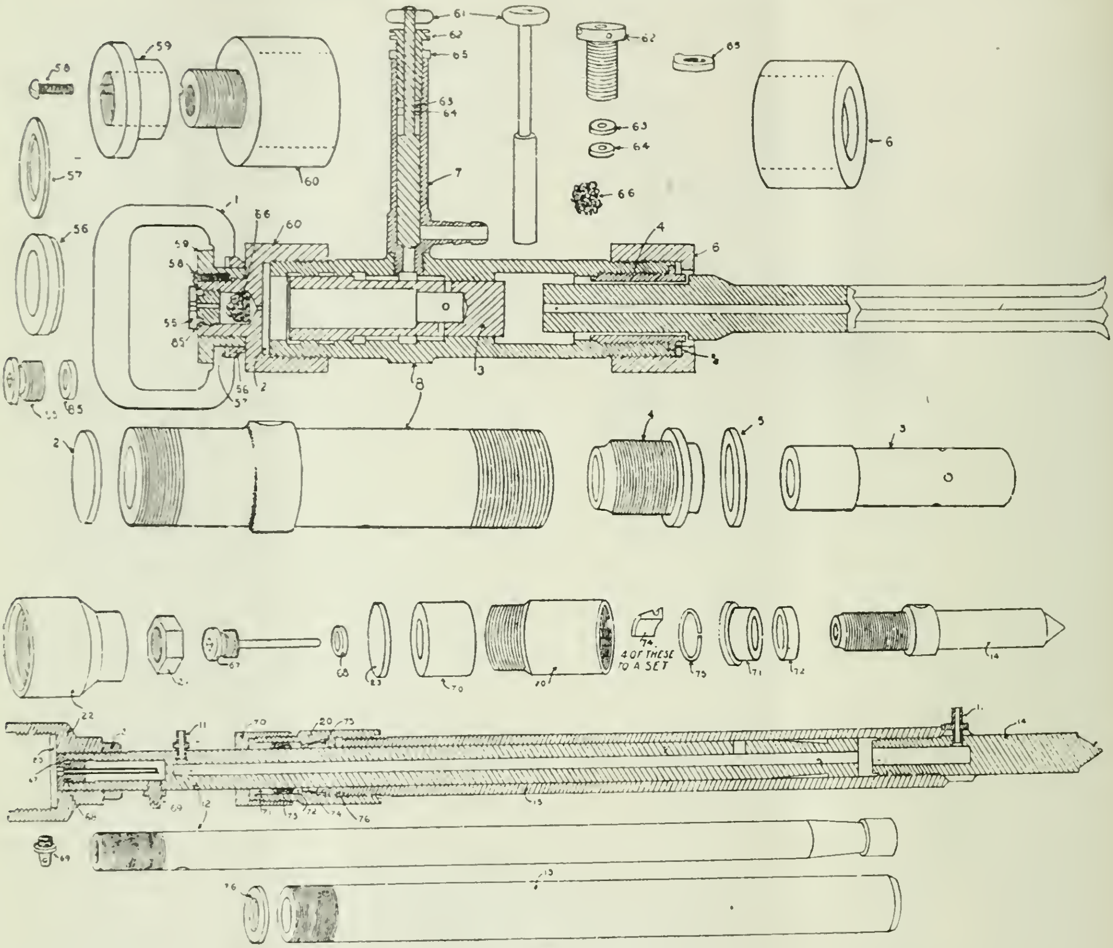

Fig. III.—Murphy Drill.

1. Handle.
2. Leather washer for handle.
3. Hammer.
4. Bushing.
5. Leather washer for bushing.
6. Cap.
7. Valve casting.
8. Cylinder.
11. Tool steel nipple.
12. Piston rod.
13. Cylinder.
14. Tail piece.
20. Feeder head.
21. Feeder lock nut.
22. Feeder cap.
23. Leather washer.
55. Oil plug for handle.
56. Rubber buffer.
57. Steel washer for handle.
58. Lock screw for handle.
59. Nut for handle.
60. Back head.
61. Valve stem.
62. Stuffing box gland.
63. Stuffing box ring.
64. Stuffing box ring.
65. Lock nut.
66. Sponge.
67. End oil plug.
68. End oil plug washer.
69. Side oil plug.
70. Feeder stuffing box nut.
71. Stuffing box gland.
72. Stuffing box ring.
73. Packing.
74. Dogs.
75. Spring for dogs.
76. Leather washer, feeder head

a success elsewhere a failure here? In trying to answer this question I must borrow largely from current American technical literature. The working air pressures on this field are only 40 to 75 lb. per sq. in. These machines were all made to be operated by air at 90 to 100 lb. pressure. G. Leyner, the pioneer of hammer drills, is reported to have said that it is impossible to get satisfactory work from a hammer drill with air pressures below 70 lb. The reason of the failure of these hammer drills is now obvious, apart from certain other defects. They were all provided with a light piston weighing $\frac{1}{2}$ lb. to 1 lb., with a very short stroke, and they relied for the kinetic energy of the blow (the product of half the mass and the square of the velocity $\frac{MV^2}{2}$) on the high velocity caused by the high air pressures, first to overcome the inertia of the steel, and secondly, to cut the rock; and they also relied on striking 1,500 to 2,000 blows per minute. Here the rock wanted more cutting, and, owing to the low pressure there was no velocity on the hammer, thus they could not bore well; the number of blows per minute was reduced also. Then all the other drill operators, except those connected with the Gordon drill, were troubled with the supply of suitable hollow steel, and even the latter had some trouble, I believe. G. Leyner at first had to use welded bits that were always breaking, and the steel sent from America for his small machine and also for the Hardsoeg Wonder machine, though reported to be satisfactory there, was almost useless here. Both these machines are hung up at present, partly for this cause, though both makers would, I think, have to lengthen the stroke on their machines before they could be adapted for use here. Then no machine tried, except the water Leyner, had any suitable water feed attachment for use with hollow steel, and here a hammer drill is useless without water being led through hollow steel to the cutting edge of the drill bit. Fig. IV. is a sketch showing the arrangement of the front head, water feed device and anvil block of the Gordon drill. C is the cylinder wall, H is the striking hammer, A is the anvil block, F H is the front head in one piece: this latter has to take up and resist the blows of the hammer should the steel not do so. W R is a hollow water ring secured by the ring R and the lock nut L N. S is the shank, and O the stem of the hollow drill bit. Air is exhausted through A, and meets the water coming from the ring along the hole P, and carries it down the hollow steel. The designers of the Gordon drill were in a position to understand the bearing these failures had on the solution of the problem. They decided to use a stroke of 1 in., and a piston weighing 2 lb., and to strike a heavier blow, relying more on the mass than on the velocity (which after all must be increased four times to double the kinetic energy) for the cutting force of the blow. Without an anvil block such a blow might have given trouble in burring up the steel, and in their system of water feed an anvil block is a necessity, hence an anvil block is used. Heavy blows are, however, very much severer on the steel, and on the anvil block and on the machine itself, especially in hard ground. It is claimed that suitable steel to withstand these stresses has been procured. The makers of the Murphy drill were also not discouraged by their first failure, and they claim also to have produced a water feed machine of simpler construction than the Gordon drill, and one that will bore twice as rapidly as their former machine did at 60 lb. pressure. Some of these drills are due to arrive shortly, and other American makers and also an English maker are also adapting machines for this market, so that we may be sure of getting a variety of types. When I speak of boring at 40 or 50 lb., I quite recognise that it does not pay to work any machine either of hammer or piston type at this pressure, and I believe that air pressures will have to be improved all round if any type is to be a permanent success.

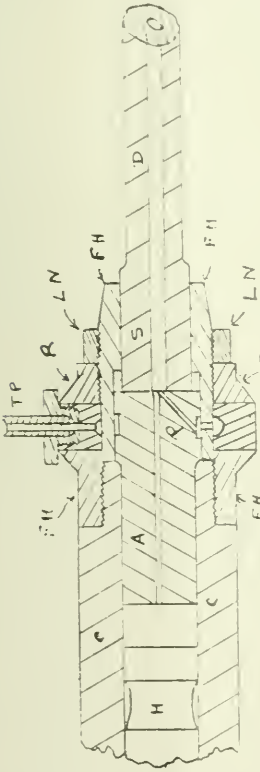

Fig. IV.—Gordon Drill.

In turning to a comparison of hammer and piston drills, we must now ask why the hammer drill beats the $2\frac{1}{4}$ in. piston drill stoping in America with air pressures of 90 or 100 lb. In Cripple Creek, the rock is mostly fairly hard granite intersected by, I think, phonolite dykes. This rock cuts better than our quartzites and conglomerates. A miner experienced there told me that the general difference of gauge between piston drill steel was nearer $\frac{1}{8}$ in. than the $\frac{1}{4}$ in. used in hard ground here. All holes drilled in stopes there are almost vertical up holes. The law does not bother about dust production: the miners manage to wear respirators, or die. Then, even at 100 lb. pressure, the hammer drill does not bore as fast as the piston drill: but owing to the introduction of the air feed as shown in drawing (Fig. III.) no arm bar or clamp is required for these 2 in. diameter hammer drills. No time is lost in rigging up, as the tail of the air feed is placed on a board in the stope filling, and drilling is started within 15 minutes of entering the stope. It is well known that only about 50 per cent. of actual working time is employed in boring with piston drills, the rest being taken up in rigging and adjusting machines, and in changing drill bits, etc. In boring under these conditions with hammer drills most of this time is saved, hence better work is done. In an eight-hour shift, 40 ft. is bored by a 2 in. Murphy drill, against 25 ft. with a 2¼ in. piston drill, even though they do not bore as rapidly when actually running. Repairs are cheaper than with piston drills, working out at about half. The air consumption is less per foot drilled, though not much less per shift, as the hammer drill is, as before pointed out, working more continuously. Solid steel is used in these up holes, hence in this class of work the cost of drill steel runs about the same as with piston drills.

What are the conditions here? Are they equally favourable, and are they such that we can expect the same advantages from hammer drills? Here the rock is harder to cut, and heavier blows must be delivered on hollow steel, which is itself more expensive in first costs—cost of upkeep, and cost of sharpening, than the solid steel used in most piston machines. No welding can, I think, be permitted. The hammer drill cannot be used with the air feed alone, as all stopes are inclined, though it might be possible to work steep stopes at Randfontein and elsewhere with sand filling on this method; but it is doubtful if the water feeds would not give trouble. Here then, as in a piston machine, a bar with some kind of arm and clamp must be employed, hence not much time is saved here in this respect and not much time is saved in changing bits. A water feed with its great advantage and corresponding expense must be used, and nearly all the holes will be bored a few degrees on each side of the horizontal.

As far as I can learn from others, and from experience in mining with one of the smaller types of hand hammer drills, some of the disadvantages of this type are that, as the holes drilled have very little clearance, great care must be taken in alignment, or the hole will get out of truth and be lost. Even in boring vertical and up holes the drills are liable to stick, and in America several patent and special tools are employed to release such drills. Here in boring down holes any temporary stoppage of the water, due to grit choking cocks, or passages, is almost sure to result in a stopped drill. The same trouble is caused by a soft drill burring up on the ends, choking the central hollow core, and wedging itself by burring up its ends. The chief troubles, met with so far, have been owing to breakages, and this, of course, may be got over by improved design and material. The types of big Leyner drill, however, at work here, cost £6 to £7 per month for repairs. The advantages proved for the hammer drill, with water feed to date, are a very fair boring speed in ground of moderate hardness, capability to be handled by one operator, the power of drilling at the same rate of speed in any directions, power to work in small stopes, and absence of dust trouble. Pillars can be more easily cut out and stopes run more parallel to the levels, thus allowing rock to be shot always towards box holes. I am certain that this type of rock borer with its few wearing parts is going to be largely used here, and I am pleased that the advantages of passing water to the cutting edge of the drill bit down hollow steel are likely to be demonstrated. The Leyner people in America, claim to be beating 3¼ piston drills in footage bored per shift, and I am convinced that any success this machine and others may have had, is due to the aid thus given to boring progress. If this system can be successfully applied to piston operated machines, a great increase in efficiency will be at once shown. In attempting in my last paper to forecast the development of rock drilling on this field, I anticipated the rise of the hammer drill for short holes.

With regard to piston drills, I thought that the practice of using air expansively was scarcely likely to be in the long run economical. Since then, the drill known as the Konomax, the invention of Mr. Mauss, of the Brakpan Electric Works, has come into prominence. As its design, and the principles on which it works has never yet, I think, been brought before a technical society; I here give figures showing the general design (see Fig. V.). The piston consists of two

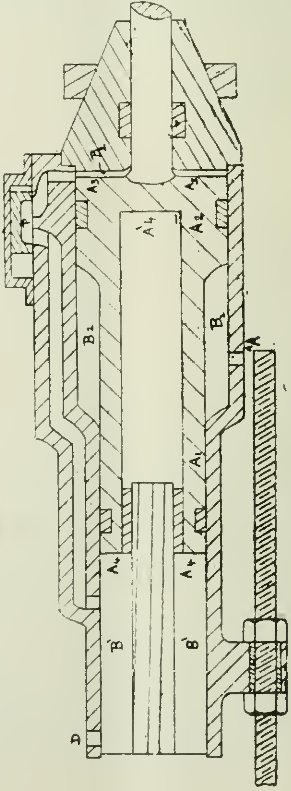

Fig. V.—Konomax Drill.

portions, A_1 and A_2, working in a corresponding cylinder. The effective area of the face A_3 being double that of the face A_4. The effective area of A_4 is much greater than that of other piston drills of similar diameter, as passages are made past the rifle bar, so that the air pressure acts effectively on the face, $A^{1}4$. This cannot be done with ordinary types, hence this machine has a somewhat more powerful blow for the same piston diameter. The air space of the smaller cylinder, B^1, is in constant communication with the air supply in the hose pipe through the inlet D. The supply of air to the fore part of the large cylinder air space B_2 is taken from B_1, and is controlled by a piston spool valve P, worked by fluid air pressure to effect the following cycles. The piston being in the position shown, live air is admitted from B_1 to the front of the piston, and acting on the face A_3 (which is double the area of A_4) starts to drive this piston back against the lesser total pressure on the face A_4. At a predetermined point in the back stroke, the valve cuts this supply off, and the air is allowed to expand. This point is generally at half stroke, giving an expansion of 2 to 1. As expansion occurs, the pressure on the face A_3 is diminished until it is only equal to that on A_4. This pressure on A_4 is then able to overcome the momentum of the piston and bring it to rest. At this instant the air is exhausted from the front end of the piston by the movement of the valve, and the piston is driven forward, and makes a free stroke. Air is again supplied to A_3, and the cycle repeated. Atmospheric pressure is maintained between the piston rings by means of an aperture A in the cylinder walls. This is in connection with the arrangements for moving the valve. The machine thus exhausts air once in the cycle and that upon the backward stroke, instead of twice as in the ordinary machine. The air is transferred from B_1 to drive the cylinder back. The pressure is always maintained behind the rear end and allows the valve P being placed right over the front end of the cylinder, thus making the exhaust part short and straight. There is thus no air lost in clearance spaces. The cold exhausted and expanded air having a direct exhaust, freezing up (which would otherwise occur) is obviated. The following is a statement of the air consumed by the Konomax drill and one of the usual type with, say, 3 in. dia. piston :

Ordinary Type.

On Front Stroke.—Piston area = 7 sq. in.; stroke = 6 in., cub. measurement of air used 42 cub. in. *On Back Stroke.*—Piston area less $1\frac{1}{2}$ in. piston rod = $5\frac{1}{4}$ sq. in., stroke 6 in., cub. in. air used = $31\frac{1}{2}$. Area of valve ports and clearance $18 \times 1 \times \frac{1}{2}$ in. = 9, therefore air used per double stroke about 82 cub. in.

Konomax Drill.

Area of A_3 (see Fig. V.) = 14 sq. in., stroke 6 in., cut off at half stroke = $14 \times 3 = 42$ cub. in.

This shows a theoretical saving of about 50 per cent. air, and tests on new machines are said to show this to be obtained. Owing to economy in the valve chest and air port weights, despite the difference of size of cylinder, the weight of the drill compares favourably with that of drills of the ordinary types. Its detractors and competitors question its power of rapid boring and state that any theoretical saving of air will be offset in practice by increased leakage. Increased leakage is bound to occur past the piston when worn, as compared with that of an ordinary machine. In such a machine each end is under pressure half its time and the total leakages surface is twice C where C = circumference of a 3 in. circle = 9·4 in. for the whole working time. In the Konomax drill B_1 is under pressure all working time = 9·4 in. leaking surface, and also the 6 in. diameter piston ring under pressure half its time = 9·4 in. Total 18·8, or twice that of an ordinary drill. This may not, however, prove very serious, or very greatly reduce the saving of air. This drill certainly presents great possibilities in air economies for piston rock drills and also for hammer drills. It has one great defect that must mitigate against its use in certain work, such as putting down long holes in shafts and winzes. The length of the stroke is determined by the gradual equalisation of two opposing forces, the air expanding on the front area and the constant pressure on the back. Any addition to the last force means that the stroke will be shortened and the drill steel be inclined to stick about this point. This must occur where the weight of long steel is added to the pressure on the back of the piston, or also in flat dry holes where there is great friction on the steel. Hence in such work the ordinary type of machine must more than hold its own. However, in general free boring, stope holes inclined below the horizontal, I think it proved that it bores about as rapidly as any drill at half air consumption, costing, say, 3s. to 4s. less per shift for power. Where I, however, think the great promise of the machine lies is in the case of those mines wallowing in the slough of low air pressures. I am pretty certain that the Konomax drill boring at 70 lb. pressure, in most cases, could beat any other drill boring at 60 lb. Take the case of such a mine throwing out all other machines and installing an equal number of this make, they should find the average air pressure improved to such a degree that efficiency in boring would be increased 20 per cent.

I have no interest whatever in this drill, and there still remains the doubt of its efficiency over long periods; but I think, if adopted in this vay, it will prove a success, provided that work and material are first class. If, however, it is ntroduced in twos and threes in large mines, it may fail, as the men prefer, I think, the old makes. What prospects does it hold out for a small stope drill? For working with long holes in 40 in. stopes I should certainly like to try a 2¾ in. Kenomax of regular pattern and would expect to save 2s. per shift in air consumption by its use.

Mr. Mauss has, however, adopted this principle to a most ingenious one man stope drill of 2 in. piston diameter. This machine is only in its trial stages, but is certainly the most novel attempt to solve the small stope problem I have met with. The valve is placed in the piston itself. The forward feed is by a hydraulic ram, and the waste water from this feed is driven down a hollow piston and hollow steel by a portion of the exhaust, to damp the cuttings in the hole. The rest of the exhaust and water form a spray from the front of the chuck. The machine is said to be incapable of knocking the front head and to work almost automatically. The air consumption is about half that of any other 2 in. machine, and there is, of course, no feed screw. Doubtless, difficulties will be met with in practice, but the whole idea is most ingenious and promising.

Fig. VI. shows the construction of the Temple-Ingersoll electric air drill, which I mentioned in a former paper. This is now, I believe, about to be introduced on the Rand. It appears to possess many striking features of simplicity and economy, and should be most useful for certain work here where mines have a supply of current available for underground work. Where drives can be run with one machine working one shift, and where ventilation is good it should prove economical, as it was only one-third of the power necessary for 3¼ in. air drill, and it might easily be taken into very flat stopes and used there.

The Bantam drill is of only 1½ in. diameter piston. In this also the valve is contained in the piston. The air is not worked expansively, and generally the design is the same as that of ordinary types. Its weight is about 56 lb.

The Kid drill, made by the Rand Drill Co., is at work on the Crown Deep. It is stated to be stoping ground for 5s. 9·4d. per ton against 6s. per ton by hand labour. The plain slide valve moved by a tappet makes the valve action very positive and has several advantages. The wear on the guides is taken up by hammering in the malleable casting: but this does not seem an altogether satisfactory arrangement.

The Holman small drill has also a 2 in. cylinder. The designers have used the ball regulating valves as in the large machines, but have discarded the air moved valve with the D slide combined, which I adversely criticised before, in favour of a spool piston valve like that on the Ingersoll machine. The chuck is said to be a much improved one, enabling unshanked steel to be used, allowing rapid changing and easy rebushing. The stroke is a long one for this size of drill, being 5 in. The weight of drill is 93 lb. and length of feed 17 in.

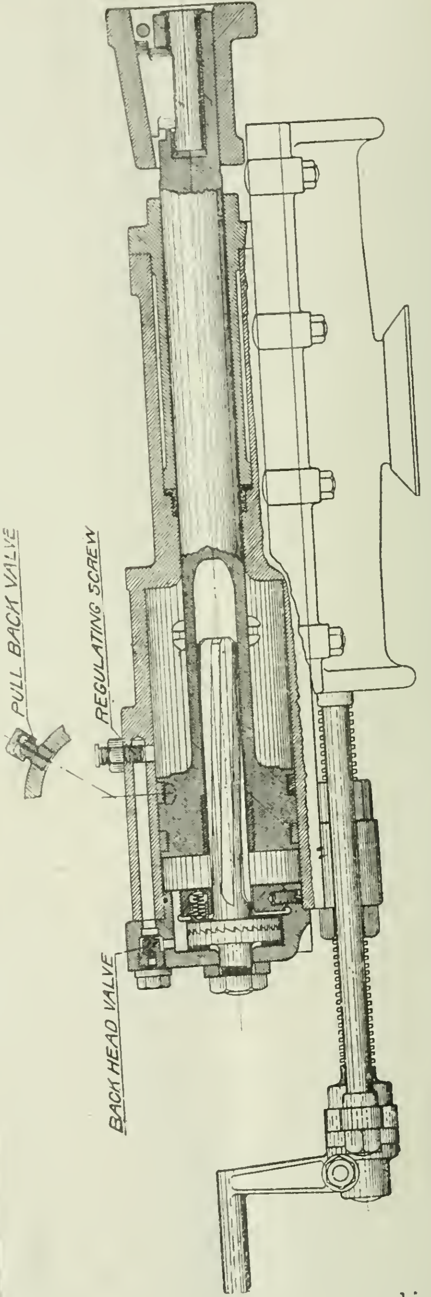

Fig. VI.—Temple-Ingersoll Electric Air Drill.

The Little Wonder and Imperial drills made by Stephens & Co. are fluid moved piston valve drills of the usual type. The chuck used on these machines is claimed to be a great improvement on the ordinary U bolt chuck in cost of upkeep and rapidity of changing. The Ingersoll Company are not pushing the sale of their 2¼ in. Ingersoll-Sergeant machine, evidently pinning their faith on the Gordon drill.

The Imperial stoping drill is the best out and out attempt to construct a one man machine. It weighs only 75 lb. and has novel features deserving notice. The piston diameter is 1¾ in. and stroke 4 in. An attempt has been made to enable drilling to be performed with equal rapidity in all directions by passing air down hollow steel to the cutting face. The benefit of this in boring up holes is obvious, as generally the bit has to cut the rock over and over again. Whether there is a perceptable increase in the rate of boring wet holes is more doubtful. Mr. Stephens, the manufacturer, has had very good results in practice, and has obtained hollow steel that appears to stand the severe work in piston drills. Recently his 2½ in. machine bored a flat hole in one-third of the time taken by a rival machine of the same diameter using solid steel. Undoubtedly it will do better and quicker all round boring work than any others at present at work, and if it will stand up to the work will, I think, be a success. The trouble with it is that a spray must be used with it when uppers are bored, and miners do not like sprays, as they do not allay all the dust, and the only water often available for them is foul water. Then the cost of upkeep of hollow steel, and of sharpening, is, of course, more than with solid steel, but a very small increase of boring rate will soon pay for that.

I still believe the real need of the Rand is for a small water feed piston drill, if these small drills are going to hold their own against the water feed hammer drills. From my remarks on hammer drills, it will be evident that I think they are largely the coming solution of the stoping problem, as undoubtedly, they should be easier to keep in repair, and cheaper to maintain, than most piston drills. The causes of failure in the past are, I think, largely removable and, provided with water feed and hollow steel, they should bore almost as rapidly as piston drills with solid steel and are easier one man machines than most piston drills.

I have now on order a number of sets of hollow steel, provided with water attachment for use with any make of piston drill, and these will be sufficient, should the quality of the hollow steel come up to expectations, to prove whether an ordinary machine can be got to drill with increased speed in any direction. If these are a success, I should like to try them in a special machine designed for them.

As there would be no trouble with mudding any hole, up or down, I should like to try the effect of shortening the stroke to 2 or 2½ in. using a larger cylinder diameter. The present long stroke on machines is adapted for low velocities of piston, and to allow of the steel kicking out the mud from the hole, and also to allow of the blunted steel, which designers know is often used, crushing the rock instead of cutting it. Of course, a drill bit must get more or less blunted; but by using mostly star bits with a water feed, and working with lengths of steel only a foot in difference, it should be possible to put down a 1 ft. hole all the way with sharp steel. The long stroke tends to loss of power, owing to vibrations on the machine, causing the drill bit to strike obliquely on the sides of the hole, thus wearing the corners off the bits and destroying some of the force of impact on the bottom of the hole. The striking energy of the blow depends more on the mass moved than on the velocity, as to double the energy developed we have to increase the velocity four times. Therefore, I should think, that such a drill with a short stroke, provided it was always supplied with sharp steel, should be able to combine some of the advantages of both hammer and piston drills. By shortening the stroke, a valve or valves might be arranged to allow direct exhaust of air at both ends, and very short air entrance ports. With air from 70 to 90 lb. pressure, the number of blows per minute should be run up well above 900, the rock being then cut by a number of lighter blows. Such a machine would not be of much use with pressures under 60 lb., and no use at all where any drill bits had to be used twice; but with high air pressures (which will surely be arranged for in all new mines by putting up compressors to work at 100 lb. or over, and by scientifically designing the underground pipe lines and putting them in under supervision with the best material and labour) it should make an ideal stoping drill, as a simple arrangement could be devised to make it easily controlled by one man. It would avoid the trouble of needing great care in alignment that hammer drills have, and no bits should stick, even if the water feed were cut off. The stroke being shorter, the edges of the steel would be less worn, and steels could be run on a smaller difference of gauge. Working with sharp steel, and a shorter stroke, there would be much less wear and tear, as it is pounding blunt steel on the bottom of a hole, that knocks piston machines out, apart from carelessness in starting holes. This is, of course, only my idea founded on, at present, an unproved hypothesis, that a water feed can be successfully used in a piston drill for a long period under working conditions, and my reasoning might prove quite wrong in practice, and such a machine not bore at all rapidly.

Really, after all, we know but little of the science of rock cutting and boring here on the Rand. With all our boasted "brains to the square inch" we have worked disgracefully in the past and wasted power wholesale. Are we going to be content in the future to leave the solving of this problem to manufacturers who very likely have no proper data to go on themselves? We have thousands of horse-power applied to boring rock, and yet we are working very much in the dark regarding the true economy of the process. For instance, we know it cannot pay to use a drill bit after it has been blunted to a certain degree; yet thousands are used twice, and we really know nothing of the amount of loss in boring, say, a foot with blunt steel against a foot with sharp steel.

In mechanical engineering the exact angle of the cutting bit is calculated for different work in different material. Conditions are, of course, different in rock boring; but no experiments are made to find the best angle for cutting edges of drill bits, and what difference a cutting edge of 90° makes to one of 60° in hard and soft rock respectively. Taking into consideration the extra cost of sharpening, providing, and removing extra steel, time lost in changing bits, would it or would it not pay to run steels in big machines with a difference of 12 or 15 in. instead of 18 or 24 in.? In boring average reef, what is the best ratio to start with of foot pounds of energy per inch length of cutting edge for the blow given by a rock drill? What is the best relation between mass and velocity for the striking body? For instance, would not a $2\frac{3}{4}$ in. machine drilling at 80 lb. pressure bore faster, even with the same sized bit, than a $3\frac{1}{4}$ in. machine working at 60 lb.? All these and a host of other questions are, I think, worth answering. Some few experts and some manufacturers may know all about them; but I doubt if they do. Surely, it would be worth the industry's while to have a portion of a mine set apart for testing all these questions in our own rock. It would be easy to prepare a large syllabus of experiments worth trying, amongst others to find out what is the exact increase of boring speed with increased air pressures. The various types and makes of machines could then be tested by impartial investigators and their relative efficiencies under working conditions compared.

I have not dwelt much upon relative air consumptions, for it is evident that this is a relatively unimportant question, because even the saving of 50 per cent. of air in a $3\frac{1}{4}$ in. drill means only about 3s. to 4s. per shift, and, therefore, a drill with double the air consumption is a better drill to use, if it can break only, say, $\frac{1}{2}$ ton more per shift, or, say, bore from 2 to 3 ft. more. The difference is, of course, less with 2 in. drills.

The same principle applies to cost of maintenance. How much more must a machine that costs £4 per month for maintenance bore than one that costs £2 per month? Obviously supposing rock broken costs 6s. per ton, and that a 4 ft. hole will break 1 ton, it has only to bore enough to break 7 tons extra per month, or an average of 6 in. per shift to pay for itself. Similarly, to pay for one extra labourer at 3s. per day it must break about $\frac{1}{2}$ ton extra, or bore 2 ft. more per day. Therefore, it will pay better to use a small machine requiring two labourers, using twice as much air, and costing twice as much in maintenance as another, if it will bore 4 to 5 ft. extra per shift. If the cost per ton broken is higher than this, it will need to bore less than this. I have not taken into consideration questions of capital outlay for air compressors in this discussion.

We see, therefore, that our real need is for the quickest boring tool procurable and the one that will bore the most footage in a given time will justify its existence in the long run. I think the most economical known way of breaking a 36 to 42 in. stope is by using $2\frac{3}{4}$ in. machines, boring 6 ft. holes and working with three machines and seven skilled natives under one white man, where conditions are favourable and air pressures 60 lb. and over, and with machines all working on the same face, otherwise two machines per man must be used.

The hammer drill for work in narrow stopes has, I think, come to stay. It must have, however, a water feed causing extra expense for piping and hollow steel against a lesser cost of upkeep. Whether any type will finally displace the small piston drill boring with solid or hollow steel I do not like to predict. If hollow steel is coming in for piston drills, star bits will have to be made from the solid, and I am certain this always pays in hard ground. Hence, drill sharpening machines, like the local Kimber, or that made by Word Bros., of San Francisco, which can make machine steel as well as sharpen it, must come into use more largely.

I am more and more convinced that the question of working air pressures is a vital one. I am at present employing 90 lb. pressure with $3\frac{1}{4}$ in. Ingersoll drills in shaft sinking in quartzites as hard or harder than any reef on the fields. Three 8 ft. holes are easily bored in two hours. Maintenance costs do not reach £5 per month. This shows that in stoping each machine at present on the Rand should under such air pressures bore easily 6 or 8 holes 8 ft. long per shift. One white man per machine might then be employed and costs reduced greatly. The record driving of 225 ft. per month, recently done by Mr. F. Girdler Brown, at the Cinderella Deep, was accomplished with air at 80 to 90 lb. pressure at the face.

The President: Machine drills have been so much to the front of late that this paper of Mr. Weston's appears at a very opportune moment. After hearing him to-night I think you will agree with me that the subject could not have been dealt with more comprehensively. The paper, I am sure, will in the circumstances lead to a very lively discussion: there is much debatable matter in it. Mr. Weston says that we want the quickest drill procurable. There is a great deal in that, and it brings in the question of the efficiency of drills—of which we have heard rather much of late. It may be said that there are two kinds of efficiency—efficiency from the mechanical engineer's point of view, and efficiency from the miner's point of view. Now these two points of view are absolutely different. The mechanical engineer talks of a drill being efficient when the drill uses little air, and the miner believes a drill to be efficient when it has this combined with many other qualities. As a matter of fact, the miner's definition of an efficient drill is "a drill which does its work at the smallest cost, everything included," and I think that definition was running in Mr. Weston's mind when writing his paper. A miner may prefer a drill which consumes quite a lot of air if, as Mr. Weston says, it does more work, and therefore more than pays for the slight excess of air. I hope those of you who deal with drills will come forward in the discussion. It is certainly a matter of satisfaction that there are quite a number of new drills on the market at the present moment, but it is too early to express anything like a definite opinion regarding their merits as practical mining machines: however, I hope that out of the number we have before us one or two will be found to be improvements.

A FEW NOTES ON THE REFINING OF BASE BULLION.

(*Read at May Meeting, 1907.*)

By C. W. Lee and W. O. Brunton (Members).

REPLY TO DISCUSSION.

Messrs. Lee and Brunton: Referring to Mr. Johnson's query, as to whether we found that the silver passed into the slag disproportionately to the gold, we have made the following assays, and found that in the case of bringing 670 oz. of bullion, assaying 761 fine up to a fineness of 802, that 1·529 oz. of gold and 2·46 oz. of silver was contained in the slag, and that quite 60 per cent. of this bullion was in a metallic state, having been unavoidably removed in the process of skimming

Mr. Lane Carter's remarks, on the fineness of the bullion on the Western Rand section, bear out our own experience; on the Windsor the mill gold used to assay about 925 fineness as compared with the Eastern Central section about 890, and still further east about 870. Our difficulty of base cyanide bullion has disappeared as mysteriously as it came, the bullion averaging between 880 and 890 fine. We may add that no change in the treatment during the clean-up or the smelting has taken place, so that the origin of base bullion is still undetermined.

In reply to Dr. Kirke Rose, we find that the wear of the $\frac{1}{8}$ bore clay tubes is very slight, but that they occasionally open out at the joint of the two sections.

We regret that for Prof. Wilkinson's information we had not an anemometer attached to the air supply, nor were able to note varying temperatures with the pyrometer, but must remind him that from the moment the bullion is received from the cyanide department, until the actual fine gold contents and bar are handed to the secretary, the assayer has regrettably not even time for the refreshments so urgently required, much less for taking scientific observations or working out formulæ.

NOTES ON WAIHI ORE TREATMENT.

(*Read at July Meeting, 1907.*)

By Ralph Stokes (Member).

DISCUSSION.

Mr. F. F. Alexander: Mr. Ralph Stokes' contribution on ore treatment at the Waihi mines of New Zealand opens up at least one point of considerable interest to us at the present, namely, the Barry honeycomb liner for tube mills: this method of lining a tube mill has not been tried here, as far as I know, although a complete set of frames are now ready, charged with broken silex, flint pebbles, and a mixture of Transvaal cement and sand as a setting: we hope shortly to have this set running in one of the mills on the Crown Deep.

I cannot agree with the author that there is any parallel between his description of tube milling at the Waihi and common practice on the Rand, for in referring to the Davidsen type of mill, of which we have several at work, he gives so little detail that criticism becomes most difficult from a comparative point of view. If, for example, only 80 tons per day of 24 hours is passed through the mill the comparison becomes more odious, as against our 240 to 300 tons of a coarser material. I conclude from the mention of spitzkasten underflow, that the Waihi mine is wet crushing, as with dry crushing 6 tons per hour to about 1 per cent. +60 (material used for cement), the life of a 2½ in. silex liner in constant use night and day is about two years. I mention this fact to show that what is quite suitable for one class of ore may not meet with success everywhere, and I am of opinion that a cast iron Barry honeycomb liner will not run 200 days on the Crown Deep, but may do so, say, at Roodepoort. I have often noticed when examining a well worn mill how the iron wedges (used for keying a last course) protect the adjacent flint blocks, and this led me to try several courses of grizzly bars longitudinally through the mill. The 5 in. grizzly bars wore away practically as fast as the silex blocks, and after a second trial I came to the conclusion that they were a failure: the honeycomb set I have on the mine is 15½ in. square inside, 6 in. deep, in four compartments, and 1 in. to ⅞ in thick, and, filled with silex concrete, will weigh a little over 19 tons without the ends, which, by the way, we shall not attempt, owing to the number of bolt holes tending to weaken the ends of the mill.

Had Mr. Stokes given more detail in connection with this part of his paper, such as internal diameter of mill, ratio of water by weight to solids fed, consumption of pebbles per 24 hours, rise of temperature, if any, and so on, then one could judge whether particularly good work is being done, as, for instance, the tables of grading on these fields compare somewhat as follows:—

Waihi decreases +60 from 31·03 tons to
1·28 = 29·75 = 95·87 per cent.
Crown Deep decreases +60 from 67·87 tons to
8·00 = 59·87 = 88·21 per cent.
Waihi increases −60 from 68·97 tons to
98·72 = 29·75 = 43·13 per cent.
Crown Deep increases −60 from 32·13 tons to
92·00 = 59·87 = 186·33 per cent.

It is clearly seen from the above figures that if the Waihi ore, as Mr. Stokes says, is comparable with our banket deposits there is considerable room for improvement.

Mr. Stokes' description of the vacuum filtering of slimes is most interesting, and I am sure our members would welcome a carefully written paper on this method of slime treatment, as the simplicity of the apparatus would, I feel sure, recommend itself to small concerns with high slime values.

NOTES ON THE ESTIMATION OF CAUSTIC LIME.

(Read at August Meeting, 1907.)

By Edw. H. Croghan (Member).

DISCUSSION.

Mr. W. Taylor: Before making any remarks on this paper, from the seller's point of view, I wish to thank him for the thorough manner in which he has gone into the question of the value of the caustic lime as supplied by the various lime companies to the mines. It had been thought, in the absence of any satisfactory explanation, that when the majority of the mines elected to analyse the lime by the sugar method, though the gravimetric method invariably gave much higher results, that an unfair advantage was being taken, which the lime companies, owing to excessive competition for business, were unable to resent. Mr. Croghan's exhaustive analysis now shows us that practically the same results can be reached by the two, or including the acid method, three different methods.

But in spite of the fact that these minute investigations dispose of any doubts as to the accuracy of the method in vogue for estimating the commercial value of the lime, I still think the sellers have a grievance, and a very distinct grievance, which was referred to by Mr. McArthur Johnston in his remarks at the last meeting, that is, the manner in which the sample is taken. He says, the best method is to grind 10 bags from each truck, a most excellent scheme, but one which it is extremely improbable any mine would systematically carry out.

Let me suggest a less ambitious scheme, one that has proved satisfactory in practice and occupies a very short time to carry out. On one occasion, feeling very sore at continued deductions being made, I went out to a mine and with the assayer took at random one bag of lime from a truck which had just arrived. This was thrown on to a ½ in. mesh screen, the larger pieces crushed in the sample rock breaker until all passed through, and two cross sections were taken with a shovel giving enough to fill a bucket; this amount again was crushed by hand to pass a 900 mesh screen, and then quartered down until sufficient was left to fill an ordinary assay sample tin. The whole operation took something like half-an-hour, and the final analysis proved satisfactory to all parties. If such a system of sampling were carried out thoroughly and the sample immediately "transferred to a small glass jar, the cover of which could be screwed down tight on to a caoutchouc ring, thereby ensuring complete protection from the air," to quote Mr. Croghan, there should not be the curious differences between the value of the caustic lime, as shown by the analysis and actual consumption. For instance, if a truck of lime analysing 80 per cent. CaO will keep the solutions in the desired state of alkalinity for ten days, a truck analysing 40 per cent. should surely last only five. In point of fact, further supplies are not required a moment sooner; from which must be deduced, that either great latitude is allowed in the degree of alkalinity to be maintained, or the samples are not taken with that considerate care for the pockets of the lime burners, that our cyanide managers would have us believe.

It is a well-known fact that lime absorbs moisture very rapidly, but from a commercial point of view, this seems to be ignored, the blame for inferior lime during the wet season being put entirely on the lime companies. I find, for instance, from December, 1905, to April, 1906, the lime supplied by my company averaged according to returns from several mines, 68 per cent., whereas from May to September, 1906, the analysis showed 76 per cent. caustic lime. I mention these figures to show that even supposing the samples are promptly and accurately taken, and the analysis carefully carried out, it does not necessarily prove that the lime produced by any particular company is either better or worse than any other lime, unless such conditions, as the state of the weather, the time that has elapsed between the burning and sample taking, etc., are taken into consideration.

I must apologise for travelling rather far from the points raised in Mr. Croghan's paper, but am inclined to the view that as in assaying for gold, so in analysing for caustic lime, all parties are at the mercy of the sampler.

Prof. G. H. Stanley: I think Mr. Croghan deserves the thanks of this Society for giving us the benefit of his work on this question. His paper calls less for criticism than mere comment, there being little room for the former.

I differ, however, with his definition of the term available lime, as I think it would be preferable to understand by the term that lime which is soluble in water and therefore available for precipitating slimes, etc., *i.e.*, that which is available and not that which might become available. The carbonate, of course, is not available in this sense, only the free or caustic lime being included, the user caring nothing for insoluble compounds. It seems to me, therefore, that "available lime" or "free lime" would be the most suitable term to employ.

I should like to draw your attention to a possible cause of the discrepancies in results after allowing for every possible lime compound which could be formed under the circumstances. I think the explanation, at least in some cases, say, 1, 2, 3, 4 and 7, lies in the determination of soluble silica. Some years ago I had a number of cements and mortars for analysis, soluble silica being one of the determinations I had to make, and in the course of my analyses I found that the percentage of silica soluble in acid depended to a considerable extent on the strength and temperature of the acid employed. After this experience I looked the matter up and found that this behaviour of the silica was well known, and consequently a certain strength of acid—I believe cold 10 per cent. hydrochloric — was recommended as a standard for sole use in this determination in order that results by different analysts, or of different analyses by the same operator, might be comparable. Free lime was determined by the ordinary gravimetric method as outlined by Mr. Croghan, after calculation. These variable soluble silica results were due to some of the silica from the decomposed silicates remaining insoluble, although the lime with which it had been combined went into solution and formed part of the total lime determined. I see the same fact referred to at the beginning of the paper, and it seems possible that in his analyses the same thing occurred, and that therefore his percentage of soluble silica obtained does not represent the full amount of silica which ought to be calculated to silicate and subtracted from total lime, thus leaving the free lime result too high.

No query can be raised as to the propriety of calculating CO_2 and SO_3 to calcium carbonate and sulphate, and it seems probable that the most basic silicate of the series would also be formed. The figures presented certainly lend colour to the view that an aluminate is also formed, though whether the formula given is correct appears to be a moot point—it is certainly a curious coincidence that three out of four samples, *i.e.*, 5, 6 and 9, which give the highest comparative results by the sugar method should also be the highest in alumina. I may point out in this connection that the words "high" and "low" in the third table on p. 39 appear to be transposed, and that the printer appears to have made another slip with reference to the relative insolubility of magnesia on p. 41, and that lastly the author is apparently responsible for a slip on p. 37 in stating that manganese carbonate becomes converted to MnO_2: should it not be Mn_3O_4?

Dr. J. Moir: One of the benefits which will have accrued from this paper is, I imagine, the illumination thrown on one of the dark places of analytical chemistry. To those of us here who have, so to speak, "seen the light" regarding the problem involved in determining caustic alkalinity available for protecting cyanide, it seems positively marvellous that a process so unscientific as the differential gravimetric one should ever have found anyone to believe in it. It is, therefore, the more satisfactory to find that Mr. Croghan has worked out the cause of the error—namely, the combination, during burning, of the lime with most of the impurities present in ordinary limestone.

I have my doubts, however, about his calcium aluminate—not that the formation of $Ca_3Al_2O_6$ in lime-burning is unlikely, but because I think the supposition is an unnecessary one for the present, inasmuch as the distinction which he makes between quartz and soluble silica is perhaps in reality not so definite as he makes it out to be. What I mean is that the quantity of soluble silica obtainable from a particular specimen of lime is liable to variation according to the strength of acid or soda employed and the temperature and time of extraction; consequently its true value may be rather higher than that given by Mr. Croghan in each case, and the calculation of the gravimetric figures may "square" with the direct methods without assuming lime to be bound up as aluminate. Of course, if the author did succeed in dissolving every possible trace of silica, then I am wrong, and some such supposition as that of the existence of $Ca_3Al_2O_6$ becomes necessary; but, for the present, the small discrepancy between theory and experiment, on account of which he makes this supposition, seems to me to be more likely due to the accumulation of experimental errors than to the real presence of $Ca_3Al_2O_6$ in the original lime.

My chief reason for addressing you, however, is to call your attention to the fact that in this matter of the estimation of *caustic* alkalinity we seem to have got ahead of the rest of the world by a few years. I am led to this conclusion by reading a paper on this subject published in *The Analyst* for September, 1907. The author is Mr. Jas. Hendrick, F.I.C., B.Sc. (lecturer in Agricultural Chemistry at Aberdeen University), and his paper is practically a rediscovery of the application of the sugar method brought to light here by Mr. Crosse a number of years ago, and in daily use in many Transvaal laboratories. The preamble of Mr. Hendrick's paper is practically the same as that of Mr. Croghan:—

"Methods of estimating rapidly and accurately the caustic lime in burnt limes and other substances are not given in the ordinary text-books, and do not appear to be in common use. In the past, analyses of limes have generally been 'complete' analyses, in which the caustic lime has been determined by estimating the total lime and certain other constituents of the sample, and assuming that the lime which is left after subtracting from the total what has been calculated to be present in certain compounds is caustic lime. This indirect method is liable to great inaccuracy. For instance, a considerable amount of lime is sometimes present as silicate in burnt limes. It appears to be common to include such lime in the caustic lime. It is probable that estimations of caustic lime in this way are generally in excess of the truth, especially in the case of impure limes prepared from limestones which contain a considerable percentage of fine silicious matter. In any case, such a complete analysis is usually not wanted, and is time-consuming and troublesome." . . . "The amount of calcium oxide present in a mixture of the oxide with calcium carbonate can be estimated by direct titration with standard hydrochloric or nitric acid, using phenolphthalein or phenacetolin as indicator (Sutton, *Volumetric Analysis*, 8th edition, p. 75). In the case of commercial limes this method is rendered only approximately accurate by the presence of magnesia, of phosphates, and of iron and aluminium oxides, which render the end-point indefinite. With experience, an approximate estimation can be made in this way which is probably as accurate as those made by the complete analysis method, and, like it, errs on the side of excess. This method has been used to a large extent in comparison with the one described.

TABLE I.

Caustic Lime.		Total Lime.	Total Magnesia.	Total Silica.
By Direct Titration.	By Sugar Extraction.			
Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
49·5	48·8	—	—	—
57·5	54·7	71·9	1·6	14·8
67·2	66·0	76·5	5·5	5·5
75·5	75·5	—	—	—
79·1	78·3	—	—	—
84·9	82·4	88·0	5·1	2·4
86·8	85·2	90·1	2·2	1·2

Methods in which solutions of ammonium nitrate are used to extract caustic lime are employed to some extent on the Continent. G. Berju and W. Kosinenko (*Landw. Versuchs-stationen*, 1904, *60*, 419) investigated these, and suggested some improvements. One great defect is that a solution of ammonium nitrate dissolves calcium carbonate to a certain extent. Another is that the lime cannot be estimated volumetrically directly after extraction, but must be precipitated as oxalate."

"It occurred to me that the lime might be extracted by a solution of ordinary sugar, and directly titrated in such a solution by standard acid. After I had commenced work on these lines it was pointed out to me that a method similar in principle had already been published by W. E. Stone and F. C. Scheuch (*Journal of the American Chemical Society*, 1894, *16*, 721 ; *Analyst*, 1895, *20*, 19). I have, however, as the result of my experiments, made certain alterations in the procedure of Stone and Scheuch which materially enhance the value of the process." . .

"In my experiments with pure lime and carbonate of lime, Iceland spar, and what was purchased as pure precipitated carbonate of lime, were used. The Iceland spar was practically pure calcium carbonate, but the precipitated carbonate, after drying contained only 98·9 per cent. of calcium carbonate. Lime and mixtures of lime and carbonate were prepared by heating these substances over the blowpipe. The amount of caustic lime present was estimated from the loss of weight after ignition, and by complete titration with standard acid and methyl orange.

"The solubility of calcium oxide in presence of calcium carbonate was studied. For each gram of the orginal carbonate of lime taken, 100 c.c. of a 10 per cent. solution of sugar was used. After the carbonate had been ignited, it was shaken with the sugar solution for at least four hours in a Wagner rotary shaker. It was then rapidly filtered and titrated with standard acid. The results are shown in Table II.

Table II.

(a) Iceland Spar.

Calculated from Loss of Weight.		Found by Titration of Sugar Solution.
Calcium Carbonate.	Calcium Oxide.	Calcium Oxide.
Gram.	Gram.	Gram.
1·000	0	Mere trace.
0·777	0·189	0·187
0·301	0·451	0·488
0·020	0·549	0·547
0·010	0·555	0·557
(b) Precipitated Calcium Carbonate.		
1·000	0	Mere trace.
0·534	0·341	0·325
0·044	0·535	0·517

In the case of Iceland spar, the agreement between the estimated and found calcium oxide is as close as could be expected. In all caustic lime estimations, unless special precautions are adopted to exclude carbon dioxide, there is a small loss due to the formation of carbonate. For the purpose in view, such minute losses are of no consequence, as they will in any case be less than those which are liable to occur in the commercial handling of the lime.

With precipitated lime the agreement is not so good, differences occurring which are by no means negligible. It was found that the 'pure' precipitated lime contained about 1 per cent. of silica, which, on ignition, enters into combination with a part of the lime, and prevents its solution in the sugar. This helps to explain what takes places in the case of commercial limes, which often contain considerable percentages of silica.

"*Strength of Sugar Solution.*—The effect of the concentration of the sugar solution was investigated. For this purpose a high grade commercial line, of which a complete analysis had been made, was used. In each case 5 gm. of lime and 500 c.c. of the sugar solution were taken. The solutions were shaken side by side for four hours. The results are given in Table III.

Table III.

Strength of Sugar Solution.	100 c.c. required N. HCl.	Calcium Oxide.
5 per cent.	24·5 c.c.	0·6860
10 ,,	29·5 ,,	0·8260
20 ,,	29·6 ,,	0·8288
30 ,,	29·7 ,,	0·8316

The figures show that whilst 5 per cent. of sugar is too low a concentration, there is little difference in the results obtained from the stronger solutions.

"*Amount of Shaking Necessary.*—In the method described by Stone and Scheuch it is recommended that the lime be shaken with the sugar solution for 20 minutes. I have not found this sufficient for complete extraction. Experiments were made with two samples of commercial lime, one of high quality and the other of poor quality. These were finely powdered and shaken with a 10 per cent. sugar solution, 100 c.c. per gram of lime, for different lengths of time, as shown in Table IV.

Table IV.

Time of Shaking.	High Quality Lime.		Low Quality Lime.	
	100 c.c. required N. HCl.	Per cent. CaO.	100 c.c. required N. HCl.	Per cent. CaO.
	c.c.		c.c.	
½ hour	29·8	83·44	21·2	59·36
1 ,,	30·3	84·84	21·7	60·76
2 hours	30·8	86·24	22·0	61·60
4 ,,	31·0	86·80	22·3	62·44
6 ,,	31·0	86·80	22·3	62·44

The results show that about four hours' shaking is necessary. The shaking-machine was driven at about 50 or 60 revolutions a minute. If the limes are at all coarse and lumpy it has been found that four hours' shaking is not sufficient. At the same time, no special precaution was taken to grind very finely, and they were certainly not ground to an impalpable powder.

It is most important that the shaking should be started immediately after the lime and sugar solutions are mixed. If they are allowed to stand the lime soon forms a cake of sucrate, and it is then almost impossible to obtain complete extraction. There were many erratic results obtained in some of our earlier experiments owing to this cause. . . . Even after four hours' shaking there was less lime extracted, if allowed to cake before shaking, than in half-an-hour when the shaking was commenced at once. If 2 c.c. of alcohol per gram of lime is placed in the flask before adding the lime and sugar solution the formation of a cake is prevented.

"*Effect of Magnesia.*—A sample of crystallised magnesium carbonate of the composition—MgO, 31·1 per cent.: CO_2, 31·5 per cent.: and water, 37·1 per cent. was used. It was free from lime, sulphates, chlorides, and nitrates. Five grams of the finely powdered carbonate were shaken for four hours with 500 c.c. of a 10 per cent. sugar solution, and filtered; 100 c.c. required for neutralisation hydrochloric acid equal to 0·0866 gm. $MgCO_3$, equal to 0·0412 gm. MgO. The shaking was continued in another portion for six hours, but exactly the same amount of acid was required for neutralisation. Magnesium carbonate in this form appears to be dissolved to a considerable extent by the sugar solution. A portion of the carbonate ignited to oxide was shaken with a similar sugar solution for four hours. The sugar solution was used at the rate of 100 c.c. per gram of the original carbonate taken. It required for neutralisation only 0·3 c.c. *N.* HCl per 100 c.c., equal to 0·0059 gm. MgO, or 0·0123 gm. $MgCO_3$. The oxide, therefore, is almost insoluble in sugar solution. A similar result was found by Stone and Scheuch.

It is generally assumed that all the magnesia in burnt limes is present in the form of oxide. If this be so, the small percentage present in ordinary lime will not affect the result to any appreciable extent."

"This method, therefore, furnishes a rapid means of estimating caustic lime in such substances as burnt limes, which contain carbonate of lime, magnesia, and other substances. It is more accurate than the method of direct titration, which is interfered with by the presence of magnesia and carbonate, and gives high results, and is also more accurate, as well as much less troublesome, than the method of complete analysis.

The process which I use in carrying out the sugar method for ordinary burnt limes is as follows:—

In a 500 c.c. flask is placed 10 c.c. of alcohol. Five grams of the powdered lime are rapidly weighed and added. The flask is filled to the mark with 10 per cent. sugar solution, and at once placed on the shaker, and shaken for at least four hours. A portion is then rapidly filtered into a 100 c.c. flask, and titrated with standard hydrochloric acid, using methyl orange as indicator. When the residue readily settles clear after the shaking, as is often the case, the solution can be poured off directly into the 100 c.c. flask." (See Table I. for results.)

"It will be seen that in every case but one the direct titration gives higher results than extraction by sugar. In four of these limes complete analyses were made, and the total lime, magnesia, and silica found are also given in the table for comparison.

When the caustic lime was calculated from the complete analysis, it was always found to be higher than that given by the sugar method, and generally also higher than that found by direct titration. In the case of low-quality limes which contain a considerable percentage of silica, it is not unusual to find that the caustic lime estimated from the complete analysis is from 5 to 10 per cent. higher than that found by the sugar method, unless it is assumed that a considerable part of the lime was combined with silica and this amount deducted."

This is, no doubt, a case of great minds thinking alike; it is certainly a curious coincidence at 6,000 miles distance. Since the two authors agree so closely, I shall only make a few remarks on the points where they disagree. The chief one is the strength of the sugar solution required, which Mr. Croghan puts at only 2 per cent., whereas Mr. Hendrick finds a large error when he uses a solution weaker than 10 per cent. This point can only be settled by experiment, but in the meantime I wish to point out that the ratio CaO : $C_{12}H_{22}O_{11}$, namely, that required to give the particular calcium sucrate which is freely soluble in cold water, is 1 : 6·1; consequently 5 gm. of pure lime require at least 610 c.c. of 5 per cent. sugar solution for complete reaction; so that Mr. Hendrick, in using only 500 c.c., did not apparently give the 5 per cent. concentration quite a fair trial. Though Mr. Croghan used only a 2 per cent. solution of sugar, yet by using a large quantity of it, he employed an actual ratio of 10 parts of sugar to 1 of lime, and hence probably, if he shook long enough, he got a complete enough reaction. It is also possible, of course, that the two experimenters used limes of very different solubility or fineness of grinding—anyway, more experiment is needed.

The same criticism, viz., use of too small a weight of total sugar applies to Mr. Hendrick's experiments made in regard to time of shaking required.

The other point is that whilst Mr. Hendrick filters the mixture and uses methyl orange as indicator, Mr. Croghan titrates direct with phenolphthalein. The latter process is more correct theoretically, as I shall show later on, and has in addition the advantage of making no exposure of the mixture to the carbonic acid in the laboratory air. It has, however, the drawback that the end point is unsatisfactory, and indeed hard to see at all in the case of coloured limes. Most analysts, however, do not use enough phenolphthalein in this case to give themselves a fair chance of seeing the end point. In this connection I am inclined to think that Mr. Croghan, in adopting the process of direct titration, has been lucky. I mean that by titrating in presence of the undissolved substance he compensates for the error which he has made (according to Mr. Hendrick's work) of working with a too dilute sugar solution. In fact, what has happened is that Mr. Croghan's actual process is a combination of the "sugar" and the "acid" methods: for even if he failed to convert all the CaO into calcium sucrate, yet on titrating, the end point he obtained from the unfiltered mixture in presence of acid and phenolphthalein was the true end point due to total caustic alkalinity.

Finally, I should like to make a suggestion in regard to what I believe is the only scientific way of evaluating lime as a protector of cyanide. This suggestion is, briefly, to make hydrocyanic acid itself the testing reagent. All that would be necessary would be to treat a weighed quantity of the lime with excess of *pure* aqueous HCN and then titrate the cyanide formed with silver nitrate and KI in the usual manner. I make this suggestion because I am convinced that not even the "sugar" or the "acid" method is absolutely correct. In the first place, the sugar method takes no account of magnesium oxide—which certainly has a *slight* protective action on cyanide. The argument from its insolubility is not worth much, because (1) the solubility is as much as 1 oz. per ton, and (2) if the dissolved part is being continuously used up, the undissolved part will gradually go into solution. For example, MgO is easily capable of reddening phenolphthalein (which is an acid of about the same "avidity" as HCN). Even calcium carbonate is not *absolutely* inactive as "protective alkali," though I believe it can be almost neglected in practice. It is certainly a curious accident that the indicator phenolphthalein should have originally been adopted for the "sugar" or "acid" method, since I find on investigation of the literature, that it is the correct indicator, namely, the one which most closely approaches HCN in avidity. The dissociation-constant of HCN is about $13 \div 10^{10}$ (Walker & Cormack, *Trans. Chem. Soc., London*, 1900, p. 5) whereas that of phenolphthalein is $8 \div 10^{10}$, that of paranitrophenol is $2{\cdot}3 \div 10^{7}$, that of methyl orange is $4{\cdot}6 \div 10^{4}$, that of rosolic acid is $1{\cdot}1 \div 10^{8}$, that of alizarin is $8{\cdot}8 \div 10^{8}$ and that of cyanine is $4{\cdot}2 \div 10^{7}$, whilst dimethylaminoazobenzene is the only indicator with a weaker "avidity" than HCN, its dissociation-constant being $1{\cdot}45 \div 10^{11}$ (E. Salm, *Zeitsch. f. Elektrech.*, 1906, pp 99-101). Poirrier's blue is similar to the latter, but I am unable to find figures relating to it.

It will probably be said that the method I propose is too troublesome and unpleasant for technical use. Aqueous HCN is certainly a dangerous neighbour in a close laboratory, but it has often been used in research before without risk. For this special purpose it should be prepared by action of dilute H_2SO_4 on a 20 per cent. KCN solution, passing the gas over dry 100 per cent. KCN in a tower so as to remove traces of stronger acids (such as CO_2 or cyanic acid), and collected in ice-cold distilled water. It must be kept in the dark and should sometimes be tested with silver nitrate to make sure that it does not contain NH_4CN (which is usually developed on long keeping of HCN).

The equations involved in the process I propose are:—

$$Ca(OH)_2 + \overline{2 + x}\ HCN = Ca(CN)_2 + 2H_2O + x\ HCN$$

or scientifically,

$$(OH)' + HCN = (CN)' + H_2O \qquad (1)$$

and

$$2Ca(CN)_2 + 2AgNO_3 = Ca(AgCy_2)_2 + Ca(NO_3)_2$$

or scientifically,

$$2(CN)' + Ag^{\cdot} = AgCy_2'. \qquad (2)$$

Hence 1 c.c. decinormal silver used = 2 c.c. decinormal cyanidion $(CN)'$ = 2 c.c. decinormal hydroxidion $(OH)'$ = 0·0034 gm. hydroxidion. Of course, if, despite many protests, we *must* be old-fashioned and stick to a prehistoric and misleading nomenclature, then the result is that 1 c.c. $\frac{N}{10}$ silver nitrate corresponds to 0·0056 gm. of caustic alkaline substances calculated as CaO.

The meeting then closed.

Obituary.

The death is announced with deep regret of Mr. Tom Francis, the Cyanide Manager of the Nellie Mine, Insizwe, Rhodesia.

He was elected a member on the 21st July, 1906.

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* See also this number, abstracts, p. 128, under "A Study of Indicators."

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

THE PHILOSOPHER'S STONE.*—"THE CHEMICAL ACTION OF RADIUM EMANATION, Part II." The authors have apparently discovered the "philosopher's stone" sought for by the ancients, which has the property of breaking up the "elements" into simpler bodies. The "stone" has turned out to be a gas, namely, radium-emanation, which has the atomic weight 216 and apparently belongs to the argon series. It was discovered last year that the emanation when left to itself breaks down, giving apparently lead and helium, and earlier in this year Ramsay showed that in presence of water it gives instead hydrogen and neon, without helium.

The present authors have now tried the effect of the emanation on solutions of copper salts and have obtained the marvellous result that lithium, sodium, and argon have been produced in small quantity from these materials. There is, of course, some doubt attaching to the creation of sodium since the experiments were partly done in glass vessels, but the creation of lithium from copper has now been done in four experiments, in one of which extreme precautions to avoid impurity were taken. Two other preliminary experiments have also given equally startling results, viz., (1) lead nitrate gave no lithium, but gave sodium and signs of two heavy metals other than lead; (2) thorium nitrate appears to give CO_2 as one of its degradation-products.

The following are some of the details of the careful experiment with copper nitrate: pure $CuSO_4$ was electrolysed on to a smooth rotating platinum cathode and the copper was washed with the purest water and dissolved in the purest HNO_3 and taken to dryness in platinum. This copper nitrate was dissolved and the solution divided into two similar bulbs, one for the experiment and the other for a "blank," the latter being simply exhausted and sealed: the other after exhausting received four small doses of radium-emanation in the course of a fortnight, amounting in all only to about 1·6 cub. mm. of the gas. The treated bulb became cloudy and finally green and turbid. Four weeks afterwards the experiment was stopped, the mixture frozen with liquid air and the new "copper-gas" pumped off. Analysis showed it to consist essentially of NO and argon with a little N_2, CO_2 and H_2 (origin doubtful); helium and neon were absent. The thawed liquid part was centrifugalised in a silica tube, but the greenish precipitate was mainly $Cu(OH)NO_3$. The green liquor was worked up with H_2S, and the residue on evaporation weighed 1·6 mg. and gave strong Li and Na lines with the spectroscope, and also traces of calcium. The CuS precipitate on working up showed signs of another metal—too small to identify.

The "blank" was worked up in the same way. Its residue weighed about 0·8 mg. and showed Na and traces of Ca but *no* Li. The "blank" CuS gave nothing.

The abstractor suggests, in view of these results obtained with the metals of the first group of the periodic table, that it is now quite a promising experiment to try to make gold from thallium, since both are at the head of the same group.—SIR W. RAMSAY and A. T. CAMERON, M.A., B.Sc.—*Journal of the Chemical Society, London*, Sept., 1907, p. 1593. (J. M.)

HYDROGEN PEROXIDE ON POTASSIUM CYANIDE.—"The author finds that (1) hydrogen peroxide and potassium cyanide react exothermically in aqueous solution at the ordinary temperature.

(2) The peroxide is reduced without escape of oxygen so long as residual cyanide is present, but any excess of peroxide is afterwards decomposed into water and oxygen.

(3) The products of the oxidation of the cyanide, *which do not include oxamide or oxalate*, are potassium cyanate and potassium and ammonium carbonates. Cyanate is the primary product and tends to be converted by hydrolysis into carbonate through the catalytic agency of the peroxide.

(4) A part of the cyanide is converted by hydrolysis, without oxidation, into potassium formate and ammonia.

(5) The total oxidation product (cyanate plus carbonate) accounts for about four-fifths of the original cyanide, and the hydrolysis product (formate) for about one-fifth.

(6) The curve representing the progress of the reduction of peroxide, so long as unchanged cyanide remains, is of the second order and accords with the equation $\frac{dx}{dt} = k(A - x)(B - x)$, where A is the initial concentration of the peroxide, B that of the cyanide, and x the measure of the peroxide reduced, or cyanide oxidised (*i.e.*, cyanate formed) at time t. $B - x$ is the sum of the residual cyanide and its hydrolysed product (w).

(7) The hydrolysed product (w) reaches its maximum and final value ($W = 0·2B$, nearly) before the completion of the oxidising action, and thereafter the slopes of the peroxide and cyanide curves are equal. The rate of hydrolysis is expressed by the empirical rule that $\frac{dw}{dt} = k_1(A - x)(W - w)$.

(8) If the solutions contain, at the outset, the ultimate products of the action itself, as well as fresh cyanide and peroxide, these products do not exert the influence indicated by the above equations. Hence some unknown intermediate condition probably occurs while the action is in progress.

(9) The addition of caustic potash or sulphuric acid at the outset greatly retards the whole action, but the addition of ammonia has very little effect."—ORME MASSON.—Proceedings of Chemical Society, April 29, 1907, p. 117. (A. McA. J.)

A STUDY OF INDICATORS.—"A set of seventeen solutions whose hydrogen ion concentration varied progressively from $2\text{-}N$ (strong acid) to $10^{-15}N$ (strong base) was prepared. For the concentration range, $2\text{-}N$ to $10^{-4}N$ inclusive, hydrochloric acid solutions were used. The solutions of concentrations between 10^{-5} and $10^{-11}N$ inclusive were prepared by mixing solutions of the three sodium orthophosphates in the proper proportions. Solutions of potassium hydroxide were used for the rest of the series. The hydrogen ion concentration in each solution was determined by the measurement of the E. M. F. of the H + concentration cell. The colour reaction of each solution with each of a large number of indicators was determined, and the results tabulated. The indicator solutions were prepared according to Glaser (*Indicatoren der Acidimeter und Alkalimeter, Wiesbaden, 1901*), the concentration being 0·1 to 0·15 gm. per 100 c.c. of solvent. 0·1 c.c. of the indicator solution was added to 10 c.c. of the solution to be tested. The colours were observed by transmitted light. For each adjacent pair of solutions, some indicator (frequently several) was found which gave a pronounced difference in colour for the difference in H^+-concentration existing between the two solutions. By the use of a colorimeter the author was able to determine this concentration accurately enough to calculate the dissociation constants of several weak acids. His results agree very well with Ostwald's values from the conductivity method. The dissociation constants of the following indicators were determined :—

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* See this *Journal*, Sept., 1907, p. 88.

Acids.		Alkalis.	
Methyl orange	4.6×10^{-4}	Cyanine ...	4.2×10^{-6}
Paranitrophenol	2.3×10^{-7}	Dimethylaminoazobenzene	1.4×10^{-11}
Rosolic acid ...	1.1×10^{-8}		
Alizarin ...	8.8×10^{-9}		
Phenolphthalein	8.0×10^{-10}		

In choosing an indicator for a titration, one should be taken which gives the colour change not at the neutral point but at the 'equivalent' point, that is, when an equal number of equivalents of acid and base have been used. The H^+-concentrations in 0·1-*N* solutions of the following salts are, NaH_2PO_4, 9.3×10^{-5}, Na_2HPO_4, 1.3×10^{-9}, and Na_3PO_4, 4.3×10^{-13}. The first hydrogen of phosphoric acid can be titrated with sodium hydroxide, using methyl orange as indicator, the second using phenolphthalein, and the third using trinitrobenzene. The proper indicator to use in any titration is easily determined from the dissociation constants of the acid and base." — E. SALM, *Z. physik. Chem.*, 1906, *57*, 471—501. *Journal of the Society of Chemical Industry*, June 15, 1907, pp. 642—3. (A. W.)

QUANTITATIVE SEPARATION OF CHLORIDES AND BROMIDES. — "For the quantitative separation of chlorides and bromides, the author finds that the most accurate results are obtained by the following modification of Bugarszky's method (*Z. anorg. Chem.*, 1895, *10*, 387), based upon the use of iodic acid as an oxidising agent. In carrying out the method, the quantities of potassium iodate and nitric acid, employed as oxidising agent, are so regulated that the excess of iodic acid left after the oxidation is approximately constant. Also, the quantity of substance taken, should be greater the higher the proportion of bromide present. For example, in the case of the determination of chloride in crude potassium bromide, the following quantities will prove satisfactory :—

Percentage KCl in the Crude KBr.	Weight of crude KBr. taken.	*N*/5 KIO_3 solution.	2*N*-Nitric acid.
	gm.	c.c.	c.c.
5 — 10	0·6	36	20
1·5 — 5·0	1·8	96	26
0·25— 1·5	3·6	186	35

The mixture of chloride and bromide is treated with the potassium iodate and nitric acid in a 500 c.c. Kjeldahl flask having a neck 20—25 cm. long, and the solution is diluted to 200—250 c.c. The flask is then supported at an angle of 30° from the vertical, and the contents boiled gently, until the volume is reduced to about 90 c.c. in about 30—40 minutes. 100 c.c. of water are then added, and the boiling is continued until 50 c.c. have passed off. The escaping vapour is now tested by means of a piece of fluorescein paper held across the mouth of the flask for five minutes. When the test paper is no longer coloured orange, the escaping steam is passed through 10 c.c. of a 2 per cent. solution of potassium iodide rendered faintly acid with hydrochloric acid, until the solution becomes hot. This test is stated to be extremely delicate, and when the iodide solution shows no yellow colour on viewing it against a white background, it may be accepted that all the bromine has been expelled. From 1 to 1·5 c.c. of a 25 per cent. solution of phosphorous acid are now added to the solution in the flask, without interrupting the boiling, and the boiling continued for 5 minutes after all the iodine has been expelled. The solution is then cooled, a moderate excess of *N*/20 or *N*/50 silver nitrate solution added, together with a piece of filter paper, and the whole shaken until the paper is reduced to pulp. After filtering and washing, the excess of silver nitrate is titrated with ammonium thiocyanate in presence of ferric nitrate according to Volhard's method, and then back again with silver nitrate solution.

For the determination of chlorine in crude bromine about 3 gm. of the sample (or more, if the chlorine content is below 0·5 per cent.) are added to a solution of 2 gm. of potassium iodide in 50 c.c. of water contained in a stoppered flask. After well shaking, the contents are introduced into a 400 c.c. Kjeldahl flask, 60 c.c. of *N*/5 potassium iodate solution and 24 c.c. of nitric acid added, and the chlorine determined as described above. The author draws attention to the need of precautions in taking samples of crude bromine, as the chlorine content is appreciably altered even by pouring from one vessel to another. The sample should be withdrawn by means of a special pipette from beneath the level of the crude bromine." — L. W. ANDREWS, J. Amer. Chem. Soc., 1907, 29, 275—283. — *Journal of the Society of Chemical Industry*, May 31, 1907, pp. 554-5. (A. W.)

STRENGTH OF HYDROFLUORIC ACID. — "For the determination of hydrofluoric acid, a known quantity is mixed with neutral, nearly saturated potassium nitrate solution, a precipitate of potassium fluosilicate being produced. The cold solution with precipitate is now titrated with sodium hydroxide solution and phenolphthalein, to obtain the total hydrofluoric and hydrofluosilicic acids. The whole is then heated to boiling, and again titrated with sodium hydroxide solution. From the result of the second titration, the quantity of hydrofluosilicic acid is calculated according to the equation : $K_2SiF_6 + 4NaOH = 2KF + 4NaF + SiO_2 + 2H_2O$." — A. G. BETTS, *Engineering and Mining Journal*, 1907, *83*, 753. *Journal of the Society of Chemical Industry*, June 15, 1907, p. 607. (A. W.)

THE RATE OF COMBUSTION AND PRESSURE DEVELOPED IN A CALORIMETRIC BOMB. — The apparatus used in these tests was a Berthelot-Atwater bomb calorimeter with a valve opening connected with a sensitive pressure gauge. A special top was made to minimize the resistance between bomb and gauge. Experiments were carried out with guncotton, cellulose, benzoic acid, fat extracted from fæces, and a pure vegetable proteid, edestin. The results of the investigations are summarised as follows :—

1. With an initial pressure of 300 lb., the pressures in the bomb rarely exceeded 700 lb.

2. The greater the quantity of material, the greater the pressure, although the pressure is not proportional to the weight of the substance.

3. The greater the initial pressure, the greater the maximum pressure, although regularity and proportionality are not observed.

4. The medium size capsule used in these experiments has a distinct tendency to decrease the maximum pressure, and to retard combustion. The other capsules have a much less marked, but similar effect.

5. The pellet form of material in all cases markedly decreases the maximum pressure and retards the combustion.

6. The incorporation of inert material is without appreciable effect on the maximum pressure, but markedly retards the rate of combustion.

7. Anthracite coal results in low maximum pressures, but the rate of combustion is extremely slow, sharply contrasting with results from bituminous coal.—F. G. Benedict and F. P. Fletcher.—*Journal American Chemical Society*, vol. xxix., No. 5, p. 739, May, 1907. (J. A. W.)

The Determination of the Organic Nitrogen in Sewage by the Kjeldahl Process.—(1) The direct nesslerisation of Kjeldahl digestates is greatly to be desired, but unless special precautions are taken such a procedure will result in a precipitation of the Nessler's reagent by certain constituents of the digestate.

(2) The present study indicates that under ordinary conditions no trouble will be experienced on account of calcium, no special treatment for its removal being necessary unless the calcium content of the sewage is greater than fifty times the nitrogen content.

(3) Magnesium, on the other hand, even in small amounts, will cause a turbidity in the tube and render a correct reading impossible.

(4) Magnesium can be satisfactorily removed, but to do this requires an excess of sodium hydrate equivalent to a concentration of 1·5 gm. per 100 c.c.

(5) The excess of sodium hydrate necessary to remove the magnesium is not great enough to interfere with the reading. In fact, this reading is not appreciably affected unless the concentration of hydrate is such as itself to produce turbidity, or about 0·35 gm. per 50 c.c. Nessler tube.

(6) It is not at all certain that the turbid tube results solely from the presence of magnesium. It has been determined, however, that under the conditions required for the removal of magnesium any other deleterious constituent is also eliminated.

(7) A process is recommended whereby all disturbing constituents can be removed and the Kjeldahl digestate nesslerised directly. By filtering the neutralised digestate a saving of several hours can be effected over heretofore proposed methods. — Leyland Whipple.—*Technology Quarterly*, June, 1907, p. 162. (H. A. W.)

Estimation of Tungsten in Steel.—"From 2 to 10 gm. of the steel are dissolved in about 30 c.c. of nitric acid of sp. gr. 1·2, 1—2 gm. of potassium chlorate being added if necessary. The solution is evaporated to dryness, the residue treated with 15—20 c.c. of concentrated hydrochloric acid, evaporated nearly to dryness, and a further small quantity of concentrated hydrochloric acid added. The whole is now diluted to 150—200 c.c., boiled for a few minutes and allowed to settle. After decanting off the clear liquid through a filter, the residue of silica and tungstic acid is washed repeatedly by decantation with hot 5—10 per cent. hydrochloric acid till free from iron (by thiocyanate test), and then washed free from acid with 3—5 per cent. solution of sodium nitrate. The residue and filter are treated with excess of $N/5$ potassium hydroxide solution, and heated, if necessary, to complete the solution of the tungstic oxide; after cooling, the excess of alkali is titrated with $N/5$ hydrochloric acid and phenolphthalein."—S. C. Lind and B. C. Trueblood, *Journal of the American Chemical Society*, 1907, 29, 477—481. —*Journal of the Society of Chemical Industry*, June 29, 1907, p. 695. (A. W.)

Atomic Weight of Sulphur.—The most important parts of this research may be briefly summed up as follows:—

1. A method for the preparation of pure silver sulphate was devised.

2. The specific gravity of silver sulphate, previously fused, was found to be 5·45.

3. Indication was obtained that Stas was unable to reduce silver sulphate wholly in hydrogen.

4. Silver sulphate was found to be occluded by silver chloride from solutions containing an excess of sulphuric acid.

5. It was proved that silver sulphate can be completely converted into silver chloride by heating in a current of hydrochloric acid gas.

6. 100·000 parts of silver sulphate were thus found to yield 91·933 parts of silver chloride.

7. The atomic weight of sulphur as calculated from this ratio, if oxygen is taken as 16·000 with several assumed values for silver, is:—

Ag. = 107·93 S. = 32·113
Ag. = 107·89 S. = 32·078
Ag. = 107·88 S. = 32·069

—T. W. Richards and G. Jones.—*Journal American Chemical Society*, vol. xxix., No. 6, p. 826, June, 1907. (J. A. W.)

METALLURGY.

Standardisation of Screens.*—"The committee on standardisation of screens, appointed by the council of the Institution of Mining and Metallurgy, London, has reported by its chairman, Mr Walter McDermott, that it recommends the adoption of the following standard laboratory screens:—

Mesh.	Diam. of Wire.	Size of Aperture	Screening Area
	Inch.	Inch.	Per cent.
5	·1	·1	25·90
8	·063	·062	24·60
10	·05	·05	25·00
12	·0417	·0416	24 92
16	·0313	·0312	24·92
20	·025	·025	25·00
25	·02	·02	25·00
30	·0167	·0166	24·80
35	·0143	·0142	24·70
40	·0125	·0125	25·00
50	·01	·01	25·00
60	·0083	·0083	24·80
70	·0071	·0071	24·70
80	·0063	·0062	24·60
100	·005	·005	25·00
150	·0033	·0033	24·50
200	·0025	·0025	25·00

Owing to difficulties in wire drawing and in the weaving of wire cloth, absolute accuracy to the fourth place of decimals of an inch is unattainable with uniformity of 25 per cent. of screening area; but the above table is so near to theoretical perfection, and the unavoidable irregularities of screening tests themselves are so wide that any inaccuracies in the table would be absolutely immaterial in practice. It is not possible to weave the 200 mesh screen except in what is known as 'twilled' or double wire, and it is questionable whether the table should not stop at 150 mesh as being fine enough for all practical sizing tests by screen.

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* See this *Journal*, vol. vii., April, 1907, p. 346.

The advantages of the series are :

1. That a definite ratio between aperture and mesh is secured.

2. That by adopting a screening area of 25 per cent., the wires are absolutely 'locked' in position, thereby preventing shifting and consequent irregularity in the size of aperture.

3. That the ratio between wire and aperture in all meshes being constant, the 'cup' factor is also constant.

4. That as the table is based on the curve of seriation, any other sizes of mesh can be readily determined.

The committee appends a memorandum explaining in some detail the grounds and arguments upon which their recommendations are based, and they desire to add that these recommendations are the result of the most careful consideration of the suggestions of users of screens in all parts of the world and of the views of British and American manufacturers." — W. McDermott. — *Mines and Minerals*, July, 1907, p. 554. (A. McA. J.)

Volatilisation of Gold during Melting.—"It has long been known that gold is lost by volatilisation during melting in mints, but the amount of the loss has not been determined exactly. Moissan has recently shown (*Comptes Rendus*, vol. 141, p. 977) that gold can be distilled rapidly in an electric arc-furnace, and has given reasons for believing that its boiling point at atmospheric pressure is about 2,530° C. At such temperatures specimens of gold have been found to lose as much as 50 per cent. of their weight in a few minutes. In 1893 I found that pure gold (*Journal of Chemical Society*, vol. 63, p. 714) begins to volatilise in ordinary furnaces at a temperature just below 1,100°, and that the loss per minute at 1,250° is four times as great as that at 1,100°. I also showed that the standard gold-copper alloy is more volatile than pure gold, and that the alloy loses proportionately more copper than gold, although owing to the absorption of gases or other impurities during melting, an increase of fineness may not be observable in all cases. The temperature of the gold pots in the melting house furnaces has been found to vary between 1,100° and 1,200°. The melting point of standard gold is 951°.

Assuming that the loss by volatilisation varies directly as the area of the exposed surface of the molten metal, then, judging from the results of experiments on a small scale the loss would amount to about 0·2 per 1,000 per hour at 1,200° on a charge of 1,200 oz. of gold.

The rate of loss during melting does not depend entirely on the temperature and the length of time in the furnace. The passage of a current of air over the surface of the molten metal is very deleterious in increasing the amount of volatilisation, so that if the lid of the crucible is removed for any purpose the loss is increased. The presence of volatile impurities in bullion also causes increased loss. When the bullion is melted, they pass off, carrying gold and copper with them. It is probably due to this cause that a number of pots were rejected on assay in the past year as incorrect in composition. By the methods of assay any lack of uniformity in the composition of the fine ingots used for making standard bars would be detected, but the presence of volatile impurities would not be detected.

Volatilised gold is condensed and solidified almost at once and is carried forward by the furnace gases in the form of fine dust. Where the draft impinges on any solid body some of the gold is deposited, but it is difficult to collect the whole of it. In the mint furnaces, the brickwork surrounding the flues is coloured purple with deposited gold, and a sample of 103 oz. of dust recently taken from inside the flues was found to contain 1·39 per cent. of gold. At the Sydney mint some years ago Leibius found that the sweepings taken from the coping stone of the melting house stack contained 1·46 per cent. of gold and 6·06 per cent. of silver. It is evident that in this case some of the volatilised gold must have been carried away into the air and irrecoverably lost.

In order to reduce the loss of volatilised gold, collecting chambers have been attached to the flues of melting furnaces in a number of mints. A chamber of this kind was recently built at a cost of $800 at the Philadelphia mint. In the course of little more than six months after it was built the dust collected in the chamber produced a bar containing gold to the value of over $4,500, or nearly 1 oz. in 10,000 oz. melted.

The total amount of loss of gold by volatilisation in ordinary melting furnaces is never larger and probably seldom exceeds 0·1 per 1,000 in melting a charge of 1,200 oz., but it represents a considerable percentage of the final waste after allowance has been made for the gold in the sweepings. An unrecovered melting loss of 0·1 per 1,000 would correspond to over £200 on a coinage of £1,000,000, as on an average gold is melted somewhat more than twice in being converted into coin."—T. Kirke Rose. —*Engineering and Mining Journal*, August 17, 1907, p. 297. (G. H. S.)

A Compound Assay Furnace.—"W. J. Fleck (*Mining and Scientific Press*, May 4, 1907) describes an arrangement whereby the capacity of a portable gasolene furnace may be increased materially when the cost of fuel, the expense of building, dust and slow heating operate against the building of a larger furnace. The arrangement consists of a large standard charcoal furnace erected beside the small gasolene furnace, and fired with dry wood. The charged crucibles are first placed in the charcoal furnace and heated until slow fusion begins. The crucibles are then quickly transferred to the gasolene, which is kept at a high heat, when quiet fusion begins at once, and in 10 minutes the crucibles are usually ready to pour. With this arrangement 16 assays may be melted and cupelled in a single crucible combustion No. 5 Hoskens furnace in four hours, or 50 assays in a No. 6 four-crucible furnace in the same time.

The flux used is first dried in open pans at a temperature of 200° F., and then intimately ground and mixed with the ore in a small wedgewood or glass mortar. The mixture is then transferred to the crucible, and a quick quiet fusion and a clear homogeneous slag is the result, the crucible being only slightly attacked on account of the short time of fusion. By this simple elimination of moisture and gases the need of salt or any other cover is obviated."—*Engineering and Mining Journal*, Aug. 10, 1907, p. 249. (G. H. S.)

Dangers of Ferro-Silicon.—"By reason of the death of four persons on a Swedish ship which carried a cargo of ferro-silicon the Home Office has deemed it necessary to issue a memorandum pointing out the danger of such a cargo, based on the investigation conducted by Prof. Cronquist into the disaster on board the Olaf Wijk. The opinion arrived at in course of the inquiry was that the fatalities referred to were due to gases given off by the ferro-silicon. The iron ore and quartz often contain phosphates, which in presence of carbon, and at the high temperature of the electric furnace, would, no doubt, be converted into phosphides, combining with the lime to form calcium phosphide. In the same way any arsenic present would yield calcium arsenide. These would be decomposed in the presence of water and evolve phosphuretted and arseniuretted hydrogen, both of which possess powerful poisoning properties. The explosions that have occasionally occurred are more difficult to account for, and various theories have been put forward to explain them. It seems that the presence of moisture is the cause of the accidents, and every effort should be made to ensure that the material is in a dry condition when packed, and that there is no possibility of moisture gaining access to the receptacles containing it." — *Times Engineering Supplement*, September 4, 1907. (J. A. W.)

CARBURISING IRON.—"The scarcity of manganese, which afflicts the world since the Russian disorders, has led to the invention of a very original process to carbonise iron at the end of the Bessemer operation for which manganese is usually employed. Sea-salt is now scattered on the molten metal, which augments the temperature, and gives special fluidity to the slag. In such circumstances it suffices to throw a handful of powdered coke into the metal about every four seconds, the coke being absorbed with wonderful rapidity and regularity. This does not dispense with manganese, but the amount is reduced, and results of the experiment made at Homecourt about a year ago seem very satisfactory." — *La Nature.*—*New Zealand Mines Record*, July 16, 1907, p. 551. (A. R.)

MINING.

RESCUE WORK IN MINES.* — The disaster at Courrières has been made the occasion of reviving the subject of rescue work in mines, and of again enforcing upon mineowners and medical men engaged in colliery practice the necessity of coming to some conclusions as to the best methods to be adopted in trying to reach and to remove miners imprisoned in a pit after an explosion. While after a colliery explosion many of the miners must have been killed by the shock and by the flame, as indicated by the marks of burning, and its destructive effects, as exhibited by the corpses, by far the larger number have fallen victims to carbon monoxide poisoning. When an explosion occurs there is first a deflagration or a sudden combustion attended with flame and gas. There is an enormous gaseous expansion, owing to the heat produced, and this is followed by contraction. The hot gases pass along the way of least resistance, and having exhausted the air there is created a void behind them. An explosion of firedamp and oxygen is followed by a contraction, so that when the watery vapour is condensed the volume of the gas is found to have diminished two-thirds. The explosion determines for a brief period a very elevated temperature, and this is followed by a sudden fall. Dr. Firmin Dervieux, in discussing the medico-legal aspects of the Courrières catastrophe, says that on the bodies of miners killed in explosions the effects of two different causes may be found—viz., wounds produced from without inwards and consequent upon the enormous pressure of gas at the moment of the conflagration, also wounds produced from within outwards during the cooling and contraction of the gases. These latter wounds are present only upon the bodies of dead miners found a little distance from the site of the explosion or in a gallery ending in a *cul-de-sac*. In this way may be explained the ecchymoses and multiple injuries when the wounds have been caused from without inwards and the tearing and rupture of the tissues and organs when developed from within outwards. In the blood of coalminers killed in this manner no trace of carbon monoxide haemoglobin is found on spectroscopic examination. Many of these men are killed by the column of gas suddenly striking them with all the force of a projectile. It is this shock which kills. Apart from the severity of the wounds, the proof that the miners under these circumstances are killed suddenly is the absence of carbon monoxide haemoglobin in their blood. Is it possible for absorption of carbon monoxide by the blood to occur in a dead body? I have exposed the dead bodies of animals (rats and mice) to fairly strong percentages of carbon monoxide gas for 24 hours and for as long as a week without finding any trace of carbon monoxide in the blood. The probability, therefore, is that no absorption takes place. In the Courrières mine the largest number of deaths was due to carbon monoxide poisoning. When the percentage of this gas is enormous, as it must be after an explosion in a coal mine, and poisoning occurs on a large scale, convulsions are absent. It is in the slowly induced forms of carbon monoxide poisoning that the locomotive powers of men who have been exposed to it become enfeebled. Of these slower forms of poisoning there were also evidences in Courrières, for owing to the small percentage of the gas in some places the men detecting the 'bad air' and experiencing headache and vertigo had tried to save themselves, but as the gas travelled quicker than they could, it continued to exert its evil effects. Some of the miners thus overpowered had fallen on their knees, others overcome by coma had fallen prostrate.

It appears that several of the wounded miners who were brought to the surface only lived a few days. Nearly all of them died from pneumonia either due to their having respired the hot air in the mine or as a complication of carbon monoxide poisoning. Dr. Lourties states that several of the miners died from pneumonia from 50 to 60 hours after the explosion. The inflammation of the lungs developed in each instance with great rapidity and was accompanied by high temperatures. The lesions in the lungs were extensive; they were lobar and not lobular and there was a greater tendency for the right than the left lung to be affected. The proof that some of these miners who subsequently succumbed to pneumonia had respired carbon monoxide is to be found in the symptoms they complained of. The men said they had headache, vertigo, and an overpowering inclination to sleep, while in other patients the limbs were paralysed. In some of my own experiments with carbon monoxide gas I have found animals die from pneumonia and small hæmorrhages into the lungs two and three days after exposure to the gas. Pneumonia is occasionally, therefore, a sequel to the inhalation of carbon monoxide gas." — *Colliery Guardian*, August 23, 1907, p. 353. (A. R.)

HYDRAULIC AIR COMPRESSOR.—A singularly direct method of converting water power into compressed air power for use in driving the various machinery of a mine is in operation at the Victoria Mine, Michigan. Instead of the water pressure being utilised to drive a turbine, and thereby actuate an air compressing plant, the water itself compresses the air directly without the mediation of any machinery whatever. In this way much expense of maintenance is avoided, and a far higher degree of efficiency obtained. The *Scientific American* states that a loss with a water turbine of 10 or 12 per cent., would be remarkably good efficiency, while there would be a farther loss of at least 30 per cent., in converting the power into compressed air. But in the hydraulic air compressor in question, the total loss is said to be only 18 per cent. At a point on the Ontonagon River, where there is an available head of water of 71 ft., a vertical shaft has been sunk through the solid rock to a depth of about 350 ft., where it ends in a vast subterranean chamber. From the bottom of this immense underground cavern, another tube slopes upwards to the tail race some 280 ft. above. Through the first mentioned vertical shaft the water pours in headlong torrents, carrying with it myriads of air-bubbles drawn from the atmosphere above. As the water traverses the subterranean lake in the cavern the air-bubbles rise to the surface, and the air is trapped in the domed roof of the chamber, while the water flows away up the inclined tunnel into the tail race above. It will be seen that the vast quantity of trapped air in the chamber is held at a pressure of 114 lb. to the sq. in., due to the difference in level of the water in the cave and that in the tail race. An ingenious device serves to cut off the supply of air at the intake when the pressure in the air reservoir exceeds this amount. The compressed air is led from the chamber through a 24 in. pipe to the mine, when it is distributed to the various points at which the power is required. At the Victoria Mine, there are three intake shafts, when all are working—5,000 h.p. is developed and available for use in driving the machinery of the mine. One great advantage of this system of compression is that the large amount of heat always liberated when air is subjected to pressure is carried off by the surrounding water, so that the system is practically isothermal.—*Scientific American.*—*Chambers' Journal*, August, 1907, p. 558. (A. L. E.)

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* See this *Journal*, vol. vii., Aug., 1906, p. 60; also vol. vii., Sept., 1906, p. 100.

PUMPING DEVICE.—"A very ingenious method has been adopted to extract liquids from boreholes, invented by Mr. Bruno Leinweber, of Vienna, for extraction of petroleum. It is utilised in Galicia at the deposits of the Galician Carpathian Petroleum Company in Kryg, near Maryampolski. The method consists in dipping a band or rope into the borehole, which absorbs the liquid; the saturated rope is then compressed between cylinders or rolls at the surface, to collect the oil. The apparatus utilised at Kryg was constructed under the management of Mr. Oscar Förster, at the Reichenberg Railway Workshops, Bohemia, where it is mounted on two 'I' girders for shipment; thus it arrives ready for use. The hemp band employed, 80 mm. wide and 8 thick, is similar to a mine cage cable, covered with a fibrous or plushy material to make it permeable to liquid. To a depth of 500 metres, Mr. Leinweber recommends a flat hemp band, and beyond this a steel cable. It is in three sections, jointed by stitching with shoemakers' waxed thread, which demonstrates the simplicity of the device and the facility of repairs. As the band is continuous, the only weight to lift is that of the liquid or oil on the ascending rope. At Kryg, owing to local circumstances, the apparatus is worked by hand. The amount of oil thus poured into the tank by compressing the rope is about 115 gm. per current meter. The method can also be utilised for brine, acid solutions, or even drainage of mines."—*Revue Industrielle*, July 27.—*London Mining Journal*, Aug. 3, 1907, p. 145. (A. R.)

GOOD TIMBERING AND DEATH RATES.—"Past records show that France has the lowest death rate resulting from the fall of roof in coal mines. This is due to the thorough method used in timbering gangways and working faces. The system briefly described is as follows:—Bars 10 to 15 ft. in length are supported upon posts set 1 metre (39·38 in.) apart. The bars are placed parallel to the working faces and are set 1 metre apart. They are made of iron, either an I-beam or square in the cross section, and are driven above the last cross-bar at the face, leaving about 1½ ft. of the face unprotected between the ends of the bars. As the face advances, short poles are set up to support the last bar before the next bar is put up. The miners are not allowed to work under unprotected roofs. The coal is mined and broken down by wedges, no blasting powder being used, which enables timbering close to the face."—*Science and Art of Mining*, Aug. 10, 1907, p. 370. (T. L. C.)

FERRO-CONCRETE IN MINE WORK. "Up to the present comparatively little use of ferro-concrete has been made by colliery proprietors in Great Britain, notwithstanding the extensive application of that material on the Continent and in America for supports in workings and galleries as an economical and durable substitute for timber. It is interesting to observe, however, that in the collieries of the Harton Coal Company at South Shields some underground tanks have recently been completed in accordance with the Mouchel-Hennebique system, for the collection and storage of water accumulating in the workings. The tanks in question are situated some 1,600 ft. below surface level, and their construction presented some difficult problems for solution, owing to the fact that the gallery was but 18 in. wider and 15 in. higher than the tanks themselves. In a situation such as this iron or steel tanks were not to be considered, for even if their erection were practicable, the impossibility of dealing with future leakages would render their use out of the question."—*Colliery Guardian*, Aug. 30, 1907, p. 401. (A. R.)

MISCELLANEOUS.

CORROSION OF IRON.—"Dr. A. S. Cushman, of the United States Bureau of Agriculture, discussed in an able paper the corrosion of iron. He explained why neither the carbonic acid theory nor the hydrogen peroxide theory could be regarded as adequate explanations of the rusting of iron. He presented a new theory, which considers the rusting of iron as an electrolytic phenomenon. Oxygen is not the primary cause of rusting, but hydrogen ions. The rôle of oxygen is only secondary. What really always happens when iron rusts is this: there is some short-circuited galvanic cell (due to impurities contained within and distributed throughout the iron) and hydrogen ions are set free, iron passing into solution as a ferrous ion and replacing the hydrogen. The ferrous ion is then oxidised by the oxygen in the air to ferric, and the formation of a hydrated oxide results.

All soluble inhibitors of rusting act in either of the following two ways:—Alkaline solutions prevent rusting because there are no hydrogen ions in alkaline solutions. Chromic acid and its salts, which are strong oxidising agents, prevent rusting on account of the formation of an oxygen film (not an oxide film), so that the iron becomes polarised in the sense of becoming an oxygen electrode. The author stated that a solution of potassium bichromate, not stronger than $\frac{1}{400}$ normal, will indefinitely prevent the rusting of polished specimens of metal in cold water, even if free access of air and carbonic acid is provided for. If iron in any of its forms is immersed in strong solutions of bichromate for a few hours the surface becomes passive, even after it is removed from the solution, washed and wiped.

Since the rusting of iron is the result of the existence of galvanic couples, electric potential differences exist, and the places of cathodes and anodes can be made visible by means of a special indicator which is called by the author 'ferroxyl.' Agar-agar and gelatine jellies impregnated with phenolphthalein and potassium ferricyanide show red and blue nodes on specimens of iron and steel imbedded in them: the blue nodes appear where the iron is anode and is passing into solution, while the red nodes indicate cathodes. If a certain point continues to be anode for a length of time pitting occurs. The author urged that experiments should be made to try the prevention of iron rust by means of bichromate on a practical scale."—Dr. A. S. Cushman.—U.S. Bureau of Agriculture in *Electro-Chemical and Metallurgical Industry*, July, 1907, p. 257. (A. McA. J.)

Durability of Wooden Water Pipe.—"The use of wooden water pipe, for mine and mill water supply is a subject of considerable interest to mining men, owing to the present high price of steel and cast iron pipe, and the frequent inability to secure such pipe in a reasonable time, owing to the great demand from municipalities, water companies, and others. Therefore, the following excerpts from a discussion by Mr. T. Chalkley Hatton, at a recent meeting of the American Society of Civil Engineers, on the relative merits of different types of wooden pipe are of value to our readers. Mr. Hatton's observations were based on investigations carried on during the past three years, which covered much of the irrigating work in the west, where continuous wooden stave pipe has been used for many years, and several of the municipal and industrial plants where both continuous and machine-made wooden stave pipe have been in use for at least 40 years.

His investigation of continuous wooden stave pipe led him to believe that there were too many uncertainties in its manufacture, that while the wood would undoubtedly outlast steel or iron pipe, yet its construction must be necessarily left to irresponsible, partially unskilled workmen, who in many instances had to prosecute their work under the most unfavourable conditions. He therefore turned his attention to a study of machine-made wooden stave pipe.

Here he found a wooden pipe being made, out of the same kind of materials as composed the continuous wooden pipe, and banded together by automatic machines run by capable workmen, and every stave, band, coating, and all workmanship entering into the construction of said pipe open to inspection under the most favourable conditions. The inner and outer surface of the pipe made perfect circles; the steel band, which was either of continuous band steel of any width required by the pressure to which it was to be subjected, and of about No. 16 gauge, or of steel or copper wire of any gauge demanded, was run through a bath of warm asphaltum pitch while bright and free from rust and then spirally wound upon the pipe carrying with it a heavy coating of the pitch, thus preventing the inner surface of the band from coming in direct contact with the outer surface of the wooden staves. Each end of the band was well secured by a double band around each end of the 7-ft. or 8-ft. length, in which lengths it is usually made. As soon as the banding is completed, the section of pipe is run over two rolls moving through a warm bath of asphaltum pitch, thus coating the outside surface of the wood and bands with a full protective coating of from ¼ in. to ⅜ in. in thickness. To prevent this coating from running, the pipe is immediately rolled in sawdust, which adheres to the coating and prevents it from being knocked off or abrased in handling. An extra coating can be readily applied to each section by repeating the operation, which in all instances occurring under his notice covered the bands and pipe to a thickness of nearly ½ in. This coating, as soon as cool, became very tough and could only be removed by a chisel, was not injured in shipping and handling, and came upon the work in as good condition as when it left the factory.

This machine-made wooden stave pipe, as made by one manufacturer, is provided with a wooden stave coupling about 8 in. long that fits over the spigot end, this spigot end being driven into it, making a strong and water-tight joint. The machine-made wooden stave pipe as made by the Wyckoff Co., of Elmira, N.Y., differs from this in having a tenon 4 in. in length cut on one end and a mortise of the same depth cut on the other end. This latter is in the form of a truncated cone, so that when driving the pipe home the joint becomes tighter the farther it is driven. The inner and outer surfaces of the pipe when formed by this method are uninterrupted throughout and without any projections, and the swelling of the wood by saturation of the water makes the joints as good as the body of the pipe. These pipes can be made of any diameter from 6 in. to 60 in. to withstand any pressure up to 300 lb. according to the banding and can be laid by unskilled workmen in either a wet or dry ditch."—T. C. Hatton.—American Institute of Civil Engineers from *Mines and Minerals*, July, 1907, p. 544. (A. McA. J.)

New Source of Power.—The possible future exhaustion of our coal supplies, as indicated by various Commissions on the subject, has led to the search for other sources of a supply of power. One of the most promising of these seemed to be the utilisation of the rise and fall of the tide, for in this there is a practically inexhaustible amount of available energy. Hitherto the tapping of this limitless source has seemed beyond the range of practical engineering and mechanics. And this, perhaps, chiefly because the abundance of our coal supply has prevented the attention of inventors being turned in real earnest to the subject. A practical attempt, however, is now about to be made to harness the sea in the Bay of Fundy. This well-known inlet between New Brunswick and Nova Scotia has long been famous for its tidal bore. Up this tunnel-shaped inlet, 48 miles wide at its mouth, the tide rushes with savage impetuosity. At high tide the water rises to a height variously estimated at from 42 to 70 ft., the lower figures being probably more nearly correct, and during spring tide the water in the Bay of Fundy stands 19 ft. higher than in Bay Verte, only 15 miles off; while on the opposite coast, Halifax harbour has a rise of only 7 or 8 ft. compared with some 50 ft. in the former bay. Here, then, seemed to be an ideal spot for the first serious attempt to harness the tides. A company has been formed, and it is proposed to build three dams, each

60 ft. high and 1,000 ft. long, across the river Tantramar, near Sackville. In this way reservoirs and discharge basins will be formed. Turbines placed in the dams will be set in motion as these are filled and emptied by the flowing and ebbing tides."—*Science and Art of Mining.* (T. L. C.)

Passive Iron.—"It is shown that the formation and properties of passive iron, and the process of passivifying is always electrolytic, and that when no external current is used local currents are generated from one part of the surface to another. Iron may also be rendered passive by the immersion of two electrodes of this metal in 16 per cent. sulphuric acid and the application of an e.m.f. of more than 0·65 volts. The current density must exceed a certain value for passivity to ensue. An iron rod which was being energetically attacked by dilute nitric acid (sp. gr. 1·2) became passive when the applied e.m.f. was ten volts and the current density 0·75 ampères per sq. cm. This change occurs at the anode, and may be reversed when the passive iron is made the cathode. The author describes other processes whereby passive iron may be rendered active, and concludes that the one process is the reverse of the other. After reviewing the various theories proposed to explain the phenomenon of passive iron. The author inclines to the view that the magnetic oxide theory affords the most satisfactory explanation of the observed facts."—H. L. Heathcote, *Journal of the Society of Chemical Industry.—Times Engineering Supplement*, Sept. 11, 1907. (J. A. W.)

Deflocculated Graphite.—"Edward G. Acheson delivered an experimental lecture at the Niagara Falls Convention of the American Institute of Electrical Engineers, illustrating the properties of a recently discovered form of graphite. This graphite is produced by adding water, gallotannic acid and ammonia to unctuous graphite as produced in the electrical furnace. Under these conditions the graphite is miscible with the water. When it is mixed with water, tannin and ammonia the graphite assumes what is called a deflocculated condition, a condition of fineness beyond that obtainable by mechanical means—one approaching the molecular state. Deflocculated graphite in water has been used successfully instead of oil in sight-drop feed oilers and with chain-feed oilers. It possesses the remarkable property of preventing rust or corrosion of iron or steel. The deflocculated graphite has also been successfully employed with kerosene oil as an effective lubricant."—*Engineering and Mining Journal*, Aug. 24, 1907, p. 354. (G. H. S.)

Transvaal Cyanide.—"Last year the mines of the Rand used some 8,780,550 lb. of cyanide, valued at £344,599, and the fact would seem to promise a prosperous future for the latest infant industry of the Transvaal which is concerned in nothing less than the local manufacture of cyanide from chemical deposits. The new venture is styled the South African Cyanide Company, Ltd., has a nominal capital of £50,000, of which £35,000 in shares goes to the vendors (for nine patents), £10,000 has been subscribed for working capital, and 5,000 shares are held in reserve. The working capital, we are informed, has been privately subscribed, and the Company has in contemplation the erection of a substantial factory in proximity to the Johannesburg-Vereeniging Railway. The main point, of course, is the price at which the new product can be sold, and this, we are assured, will be at least 20 per cent. below the cost of the imported article. The promoters inform us that one group of mines estimate that a saving of £50,000 may be made by it if the local cyanide comes up to sample. In addition to cyanide of potassium, the Company contemplates the manufacture of cyanide of sodium and magnesium, soda crystals, bi-carbonate of soda, caustic soda, and sodium borate, by patent processes. If the new Company can substantiate its claim to reduce the cost of cyanide by one-fifth, and that by the local manufacture of the article from local materials, with the aid of capital furnished locally, no one will begrudge it success."—*South African Mines*, June 1, 1907, p. 267. (J. Y.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

Shaft Sinking in Difficult Cases. By J. Riemer. Translated by J. W. Brough. Price 10s 6d. net. (London: Charles Griffin & Co. 1907.)

"The present translation by Mr. J. W. Brough, Assoc.M.Inst.C.E., of an authoritative work by an acknowledged expert with full knowledge and practical experience of his subject, is a welcome addition to the library of the student, and it will be valuable to the mining engineer, at it appeals directly to those in charge of mining undertakings, giving them a review of the various methods that may be used in difficult cases of sinking and providing data to facilitate the selection of the best method for any particular case under consideration.

The book is divided into four parts. The first briefly describes what may be called hand sinking, passing in rapid review the various methods of dealing with water by barrels and giving an interesting description of under-hanging tubbing which has been practised with such success in Zollern I. mine. Part 2 deals with shaft-sinking by boring on the original method of Kind and Chaudron. Part 3 gives a complete *résumé* of the present position of the freezing process, and the author's remarks under this head are particularly valuable, as he is well aware of its limits and of the cases in which its use is advisable. The final part deals with the sinking drum process, discusses the probable explanations of the cause of cracked drums, the Pattberg compound drum and the sack borer of Sassenberg and Clermont.

A bibliography of recent papers on shaft-sinking and an index have been added to the translation. The work is well printed and is provided with eighteen figures in the text and nineteen folding plates. It is an undoubted acquisition to the literature on the subject and is worthy of careful perusal and study by all mining men."—*Colliery Guardian*, Aug. 23, 1907, p. 355. (A. R.)

Electricity in Mining. By Sydney F. Walker. Price 9s. net. (London and New York: Harper Brothers.)

"In this book the author has evidently addressed himself to the mining engineer. Mining engineers in the past, with the caution which arises from the nature of their calling, have been somewhat chary of the use of a power with which they were not closely familiar. There has been a marked improvement in this respect within recent years, but there still remains plenty of scope for works on the subject. In seven chapters Mr. Walker deals with the principles and terms of electrical practice: signals and telephones: lighting apparatus: the generation of power: distribution: application to driving machinery, etc.: and faults in electrical apparatus. The book is illustrated by upwards of 250 drawings and photographs, and is excellently printed." — *Colliery Guardian*, Aug. 23, 1907, p. 356. (A. R.)

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 390/07. W. E. Kimber. Improvements in rock drilling machines. 7.9.07.

(P.) 391/07. G. Bonatto. Improvements in safety catches for mine cages and other hoisting apparatus. 7.9.07.

(P.) 392/07. J. Hosking. Improvements in and relating to rocks drills. 9.9.07.

(P.) 393/07. J. Hosking. Improvements in and relating to ore crushing mills. 9.9.07.

(P.) 394/07. A. L. Marthinsen. Improvements in the driving of gravity stamps. 10.9.07.

(P.) 395/07. G. Nocosia. Improved artificial stone and the process of manufacture thereof. 10.9.07.

(P.) 396/07. E. M. Hewlett. Improvements in electric insulators. 13.9.07.

(P.) 397/07. W. C. Boyd. Improvements in the fixing of collars, bosses and other similar pieces to shafts or their equivalent by means of set screws and the like. 14.9.07.

(P.) 398/07. J. H. Rowe. Improvements in the treatment of crushed ore products and apparatus therefor. 14 9.07.

(P.) 399/07. J. van Waart. A new device or apparatus for the production of power or energy (motion created by gravitation). 29.6.07.

(P.) 400/07. J. van Waart. A new device or apparatus for the production of power or energy by magnetism. 29.6.07.

(P.) 401/07. J. van Waart. A new device or apparatus for the production of power or energy. 29.6.07.

(P.) 402/07. T. Rasmussen. Improvements in rock drilling machines. 16.9.07.

(P.) 403/07. T. Rasmussen. Improvements in rock drilling machines. 16.9.07.

(P.) 404/07. T. O. Bryne. Improvements in internal combustion engines or motors. 16.9.07.

(P.) 405/07. R. H. Harriss. Improvements in disintegrating diamond bearing matter by chemical methods. 16.9.07.

(P.) 406/07. H. Goddard. A new and improved sun and mosquito screen. 17.9.07.

(P.) 407/07. J. G. Laidlaw. Improvements in the manufacture of artificial compressed stone. 18.9.07.

(C.) 408/07. J. H. Holman (1), J. M. Holman (2). Improvements in or connected with rock drills. 20.9.07.

(C.) 409/07. A. C. Bartlett. Apparatus for preventing the escapement of dust caused by mining drills of the pneumatic type. 20.9.07.

(P.) 410/07. J. E. H. Grose. Improvements in rock drilling machines 21.9.07.

(P.) 411/07. E. H. Martin (1), B. Chew (2.) Improvements relating to ore stamps. 21.9.07.

(C.) 412/07. A. R. Wojiechowski. An improved process and means for disintegrating and rendering friable diamond bearing grounds. 25.9.07.

(P.) 413/07. W. Wamiss. Improvements in rock drilling machines. 26.9.07.

(C.) 414/07. T. L. Lockhart. Improvements in the extraction of gold and silver from their ores. 26.9.07.

(P.) 415/07. C. E. D. Usher (1), Adair-Usher Process, Ltd. (2). Improvements in the treatment of sands or the like with liquid. 26.9.07.

(P.) 416/07. W. Burgess. Improvements in means for neutralising noxious fumes produced in blasting. 26.9.07.

(C.) 417/07. T. Parker. Improvements in and relating to the production of fuel. 27.9.07.

(C.) 418/07. N. Ceipek. Improvements in the manufacture of an improved safety explosive. 27.9.07.

(C.) 419/07. W. T. Smith. Improvements in and relating to process of extracting metals from their ores. 27.9.07.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

Adeney, G. B., *l/o* Germiston; Lancashire College, Manchester.
Clarkson, R., *to* New Rietfontein Estate G. M., Ltd., P. O. Rietfontein Mines.
Collings, B. I., *l/o* Johannesburg; P. O. Box 16, Salisburg, Rhodesia.
Durham, B. W., *l/o* Johannesburg; P. O. Cleveland.
Guest, I. A. M., *l/o* Rhodesia; P. O. Box 1, Klerksdorp.
Hellmann, Fred., *l/o* East Rand; c/o Messrs. F. Huth & Co., 12, Tokenhouse Yard, London, E.C.
Kline, R. C., *l/o* Mexico; La Jolla, San Diego Co., Cal., U.S.A.
Kyle, J., *l/o* Cleveland; P. O. Denver.
Leyson, Wm., *l/o* Boksburg; P. O. Box 74, Barberton.
Macqueen, W. P. O., *l/o* Johannesburg; Crown Deep, Ltd., P. O. Box 102, Fordsburg.
Mannheim, H. C., *l/o* Salisbury; P. O. Box 445, Bulawayo.
Pedrini, J., *l/o* Switzerland; P. O. Box 53, Krugersdorp.
Peirce, A. W. K., *to* Victoria Falls Power Co., Ltd., P. O. Box 2671, Johannesburg.
Simpson, T., *l/o* Luipaardsvlei; Ferreira G. M. Co., Ltd., P. O. Box 1021, Johannesburg.
Surmon, C. B., *l/o* Germiston; Shangani Siding, Rhodesia.
Wilson, R A., *l/o* Jeppestown; Eldorado G. M. Co., Lomagundi, Rhodesia.

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VOL. VIII. NOVEMBER, 1907. No. 5.

Proceedings
AT
Ordinary General Meeting, November 16, 1907.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, November 16th, Prof. J. Yates (President), in the chair. There were also present:—

31 Members: Dr. J. Moir, Messrs. T. L. Carter, R. G. Bevington, W. R. Dowling, K. L. Graham, A. McA. Johnston, G. O. Smart, Prof. G. H. Stanley, H A. White, Prof. J. A. Wilkinson, E. H. Johnson, W. Beaver. E. H. Croghan. J. M. Eaton, N. M. Galbraith, G. Goodwin, J. Gray, H. H. Johnson, J. A. Jones, G. A. Lawson, C. W. Lee, H. Lea, W. P. O. Macqueen, J. E. Metcalf, H. F. Roche, A. Salkinson, S. Shlom, R. M. Torin, H. Warren, and J. O. Welch.

9 Associates: Messrs. A. E. Adams J. C. Chilton, C. L. Dewar, J. A. P. Gibb, J. W. Hawthorne, J. H. Harris, R. W. Leng, W. E. Thorpe and J. Whitehouse.

11 Visitors and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as published in the *Journal*, were confirmed.

NEW MEMBERS.

Messrs. Gray and Croghan were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been duly elected, as follows:—

CHAPPELL, WILLIAM, French Rand G. M. Co., Ltd., P. O. Box 25, Luipaardsvlei. Mine Captain.

WALKER, JOHN FAIRFAX, East Rand Proprietary Mines, Ltd., P. O. Box 30, East Rand. Tube Mill Foreman.

The Secretary announced that the following gentlemen had been admitted as Associates by the Council since the last general meeting.

BRAZIER, H., Crown Deep, Ltd., P. O. Box 102, Fordsburg. Cyanider.

COCHRANE, A. C., Block A, Randfontein. Sampler.

COX, CHARLES THOMAS, East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Cyanider.

FOX, HERBERT, W., United States Reduction and Refining Co., Colorado Springs, Colorado, U.S.A. Superintendent.

MCSWEENEY, JAMES PAUL, Treasury G. M. Ltd., Cleveland. Assistant Sampler

MOIR, Dr. W. L., Denver. Medical Practitioner.

MOORE, WILLIAM ROBERT, P. O. Box 2586, Johannesburg. Foreman Mason.

NICHOLSON, ARTHUR STEELE, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Mill Foreman.

PEARCE, FREDERICK STEWART, New United G. M. Co., Ltd., P. O. Box 5, Maraisburg. Assayer.

PINTO, MICHELE, Komarovo Mining Co., Avzianopetrovsk, Gouvt. Orenbourg, Russia. Manager.

ROBINSON, SYDNEY GREENWOOD, c/o Messrs. F. L. Smidth & Co., 112, Palace Chambers, Westminster, London, S.W. Engineer.

SHED, WILLIAM B., Geldenhuis Deep, Ltd., Cleveland.

THOMAS, ALFRED MESSER, Simmer and Jack Proprietary Mines, Ltd, P. O. Box 192, Germiston. Assistant Assayer.

THORPE, WILLIAM EDWARD, East Rand Proprietary Mines, Ltd., P. O. Box 134, East Rand. Assistant Metallurgical Chemist.

The following Student has also been admitted.

TURNER, HUBERT FREDERICK SIDNEY, Transvaal University College, P. O. Box 1176, Johannesburg. Mining Student.

GENERAL BUSINESS.

The President: I move a vote of thanks to the Consolidated Goldfields of South Africa and to Mr. McArthur Johnston and his staff for their hospitality during our recent visit to their laboratory.

Mr. R. G. Bevington: I would like to second the vote of thanks. I am sure we had a most enjoyable and instructive afternoon. It makes one feel as though one wanted to go again.

The motion was agreed to.

The President: It has just been brought to my notice that Mr. Rowland, our Secretary, to-day completes his eleventh year of office as Secretary. I am sure you all hope with me that he will be with us another eleven years.

THE FILTRATION OF SANDS.

Mr. A. McA. Johnston stated that he wished to place on record in the *Journal* an experiment in connection with filtration of sands, and demonstrated the filtration with a small apparatus, using coloured water in the place of cyanide solution (see p. 153).

NOTE ON THE DAILY VARIATION OF RAND MINE VENTILATION.

By JAMES MOIR, D.Sc., M.A., F.C.S. (Vice-President).

In the course of my reply to the discussion on my last paper on ventilation, I gave an arithmetical formula for calculating at any time the total ventilation in the Ferreira Deep mine in 1904, a formula depending only on the reading of the wet-bulb thermometer at the surface in the shade (see this *Journal*, February, 1907, p. 256). This formula was $Q = 3{,}500 + 1{,}600\ (68 - w)$, and I stated that a similar formula could be got for any other mine with two nearly equal shafts, after experiments had been done to determine the new values of the constants involved.

The diagram there given shows the great annual variation of the ventilation, and the present experiments were made in order to get an idea of how the ventilation varies (1) from night to day, and (2) on rainy and rainless days, and this is what I mean by daily or diurnal variation.

It is evident that this question can be settled by obtaining a continuous record of the wet-bulb reading over a sufficient period and then finding out how much the above formula requires to be altered owing to the changes in the mine which have occurred since 1904.

This wet-bulb thermometer record is a thing which, so far as I know, has never been obtained before. The method consisted in arranging an ordinary metallic thermograph so that its sensitive part should be kept constantly moist. As the use of running water would give incorrect results, recourse was had to the capillary action of filter paper. The steel quadrant of the machine was first coated with a certain anti-rust composition of my invention, so as to prevent permanent injury to the machine, and then fitted with a quadrant of thick filter paper having a "tail" dipping into water, and the whole was exposed (without its cover[*]) on the headgear of the downcast shaft of the Ferreira Deep from Sept. 16 to Sept. 30, inclusive. I have to thank Mr. W. E. C. Mitchell for his courtesy in arranging for a suitable position for the instrument on his mine. It was fortunate that during this period the weather exhibited practically all the possible varieties which occur in a whole year, so that it was unnecessary to run the machine any longer. I should mention that the machine was first calibrated by means of a standard wet-bulb kindly lent me by Mr. R. T. A. Innes.

In addition a couple of rough measurements of the air in the shafts (by anemometer) were made to see if the 1904 formula still held. It was found that the air current was roughly 25% better than it was in 1904; and consequently the formula for calculating ventilation from the wet-bulb reading must be altered to this extent.

It is now, roughly, Q (quantity in cub. ft. per minute) $= 7{,}000 + 1{,}800\ (68 - w)$.

The actual observations were:—(1) downcast, clear day, $T = 68{\cdot}5$, $w = 56{\cdot}5$, $Q = 30{,}000$. (2) upcast, moist weather, $T = 55$, $w = 53$, $Q = 32{,}000$; but, of course, with this method of observation the error may be as much as 10%, whereas those of 1904 were got by an exact method.

The thermograph records obtained are reproduced opposite: they had to be redrawn owing to their exposure to the weather—and Table A gives an abstract of the resulting readings, and the calculations from the above formula.

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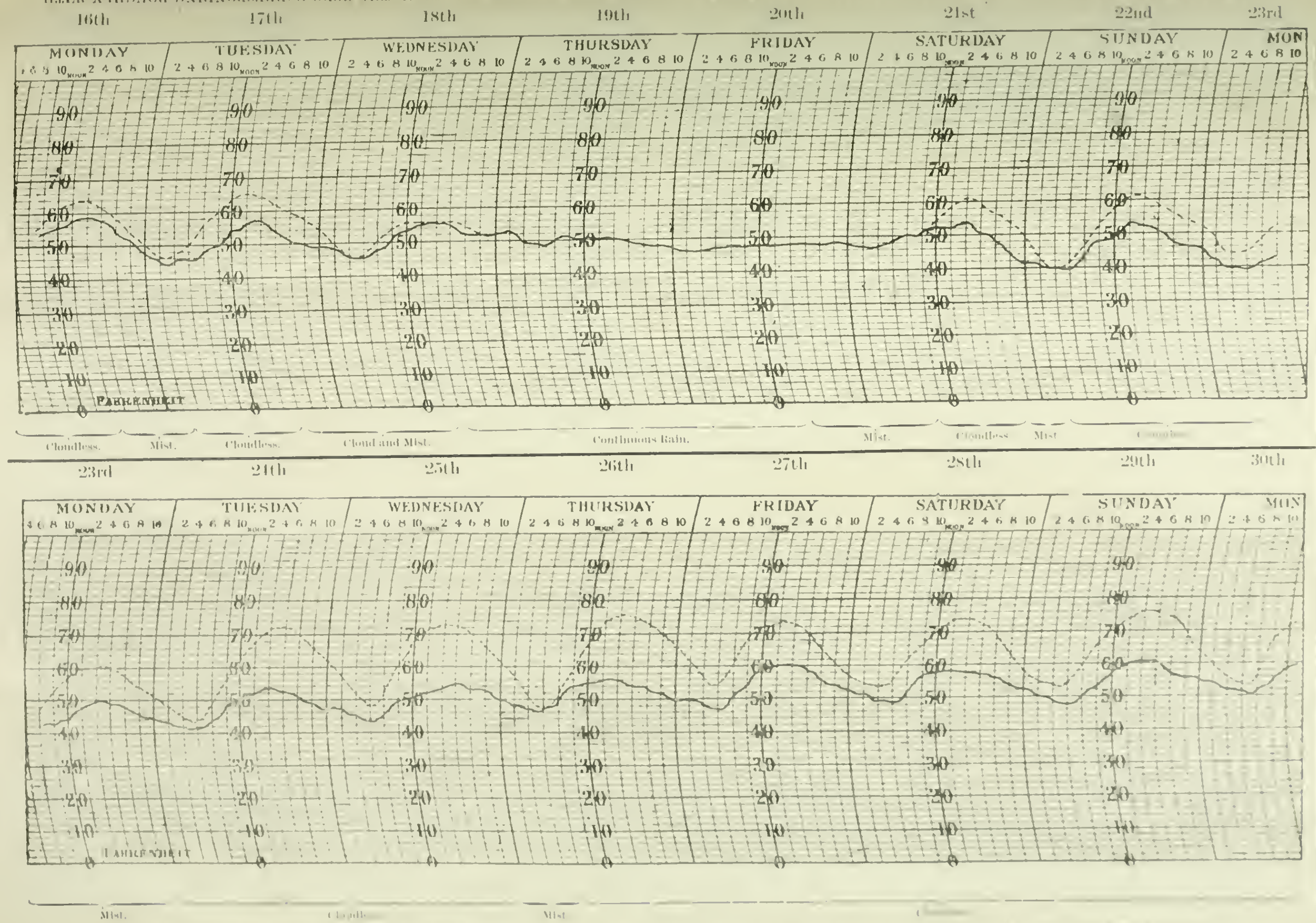
[*] It would have been better to enclose the whole in a Stevenson's screen.

TABLE A.

Date. SEPT.	Maximum depression of wet-bulb.	Minimum humidity per cent.	Minimum depression of wet-bulb.	Maximum humidity per cent.	Maxim'm distance of wet-bulb from 68°.	Minimum distance of wet-bulb from 68°.	Maximum quantity of air supply in cub. ft. per minute.	Minimum quantity of air supply in cub. ft. per minute.	Diurnal variation, in cub. ft. per minute.	Percentage diurnal variation from night to day.
16	5	67	1	93	24	9	50,000	23,000	27,000	54% less
17	9	55	1	93	24	11	50,000	27,000	23,000	46 ,,
18	3	80	0	100	23	12	48,000	29,000	19,000	40 ,,
19	0	100	0	100	20	16	43,000	36,000	7,000	16 ,,
20	0	100	0	100	22	20	47,000	43,000	4,000	9 ,,
21	7	62	0	100	22	14	47,000	32,000	15,000	32 ,,
22	8	58	0	100	28	15	57,000	34,000	23,000	40 ,,
23	11	47	4	71	29	18	59,000	40,000	19,000	32 ,,
24	19	30	1	92	26	15	54,000	34,000	20,000	37 ,,
25	18	32	3	80	25	13	52,000	30,000	22,000	42 ,,
26	21	28	0	100	22	11	47,000	27,000	20,000	42 ,,
27	14	43	7	59	21	7·5	45,000	20,000	25,000	55 ,,
28	15	40	3	80	19	9·5	41,000	24,000	17,000	41 ,,
29	15	40	3	80	20	7	43,000	19.500	23,500	55 ,,
30	16(?)	38	1	93	18	8(?)	40,000	21,000	19.000	47 ,,

LOOKED AT FROM THIS SIDE, THE THICK LINE IS A GRAPHIC RECORD OF THE CHANGES IN THE VENTILATION, THE ZERO CORRESPONDING ROUGHLY WITH



THE THICK CURVE IS THE CONTINUOUS WET BULB RECORD, AND THE DOTTED ONE THE ORDINARY (DRY-BULB) RECORD FOR THE PERIOD.

FIG. 1.—SHOWING VARIATION OF TOTAL VENTILATION DURING SEPT., 1907, IN THE FERREIRA DEEP MINE.

It will be seen that the maximum quantity of air in the whole period was 59,000 (which is much the same as the maximum observed in 1904), but it is probable (from the fact that the lowest temperature observed was 39°) that in the coldest "snap" of the year, which would be about 32° F. wet-bulb, the ventilation will now rise to 70,000 cub. ft. per minute.

The minimum observed was 19,500 this time (in 1904 it was 14,500), corresponding roughly with the general improvement due to the increased size of workings. I do not know exactly how many workers are underground at any one time—perhaps Mr. Mitchell will oblige—but assuming 650, this gives 30 cub. ft. per man per minute, the maximum being about 100 cub. ft. per man.

In addition the daily averages and the total average have been obtained by the use of the planimeter on the curves, with the following results:—

Table B.

I. Date (24 hours from 10 a.m. in each case).

	Average wet-bulb.	Average value of (68 – w) Deg.	Calculated average quantity of air.	Weather.
16—17	51·5	16·5	37,000	Morning mist
17—18	50	18	40,000	,,
18—19	52	16	36,000	Mist and rain
19—20	48·5	19·5	42,000	Continuous rain
20—21	48	20	43,000	
21—22	46·5	21·5	46,000	Dull
22—23	45·5	22·5	47,000	Clear with occasional mist
23—24	45	23	48,000	
24—25	48·5	19·5	42,000	
25—26	50·5	17·5	38,000	
26—27	52	16	36,000	Cloudless
27—28	54·5	13·5	31,000	
28—29	53·5	14·5	33,000	
29—30	56	12	29,000	

	Wet-bulb.	(68 – w.)	Quantity.
II. General average of the fortnight	50·2	17·8	39,000 cub. ft. per minute.

This is, on the previous assumption as to number of workers at a time, equal to 60 cub. ft. per head, which, I think everyone will agree, is very respectable.

With regard to the formula used ($Q = 7{,}000 + 1{,}800\,(68 - w)$) I wish to remark in order to prevent misunderstanding, that the wet-bulb is not the sole factor determining ventilation; for example, the number 1,800 in the formula really contains as its factors quantities which express (1) the square root of the depth of the shafts, (2) the section of the shaft and the rubbing surface of the whole length of the air-way.

Again, the figure 68 is intended to be the average temperature of the upcast air in this particular mine, so that if one wished to modify the formula for a deeper mine, say, 4,000 ft., one would have to alter it not only by multiplying the factor 1,800 by the square root of the ratio of the depths of shafts in the two mines, but also by increasing the figure 68 to about 78. We thus have two factors favouring increased ventilation at greater depths, as against the one contrary factor of increased friction.

The President: Our transactions are becoming a mine of information in connection with this very important subject of ventilation, and we are especially indebted to Dr. Moir for the data and results of experiments which he has carried out at considerable trouble to himself. I move a vote of thanks to Dr. Moir for his note of this evening.

Mr. T. Lane Carter: I would like to second the vote of thanks. I think these theoretical deductions agree well with our practical experience. I think Dr. Moir said he got the worst results on a very hot day. I have seen this exemplified on the Rand in a startling manner in mine air ways; where during the winter the air-current was so strong that you could hardly carry a candle, whilst in the hot December days in the same place the ventilating current was almost stagnant. This confirms what Dr. Moir has pointed out theoretically.

Dr. J. Moir: It appears that this time of the year is really the worst for ventilation, contrary to what one would expect from the analogy of other countries, where two months after midsummer would be the hottest period, and the worst for ventilation.

THE INCIDENCE OF METHODS OF PAYMENTS ON THE EFFICENCY OF MINERS.

By Kenneth Austin, M.Am.I.M.E.

In the relations between employer and employee nothing has attracted so much attention as the equitable payment for labour; the workman has to be fairly paid, and the capitalist or saver has to be rewarded and protected.

It is a well established fact, admitted by workmen and employers, that there is a considerable difference even amongst workmen when subject to the same conditions. One will waste his own time and his employer's capital, another will make the best of his time, increase the capital, and economise in every direction. It is quite right that labour should be fostered, but capital must also be guarded from loss, so that the advance of industry and of the community, can be secured with some degree of certainty.

The general condition of the wage earner has been evolved after many struggles to secure the highest possible reward for labour. The capitalists, on the other hand, have not been slow to seek those sources from which they can obtain the highest and most permanent return for capital invested. All alike are subject to the laws of supply and demand.

The object of these few notes is to state the various methods of labour payments current in the Transvaal, and to point to the evil which will arise, if at any time an artificial arbitration law should be rushed over the heads of commonsense people who have made this country their permanent home, and who will permanently feel the effects of all "class" legislation.

"Day's pay" is generally treated as if it did not involve a contract, but, of course, this is an error. The day's pay man has few or no risks, but he is always subject to the 24 hours' notice of discharge, and because of this insecurity of employment the best results are seldom obtained. In order to counteract the evil effects of this system, "bonus" schemes have been devised and tried with varying success. Universal admiration or satisfaction has, however, not accompanied any "bonus" system.

The "contractor" or piece-work man is still with us, and the writer is of opinion that this system of payment is the best, because with it both employer and employee will, under proper conditions, have the least cause for grumbling or dissatisfaction. Some contractors are useless and ought to retire from the business of "leaving the company in debt," which has been in the past their regular occupation; other contractors have followed on in the same localities as the last named, and have made money and completed contracts with full satisfaction to themselves and their employers.

Everything is bound up in the one word "efficiency." If you pay labour inadequately, the best men leave the industry: if you pay too much, you increase the difficulties with which the industry is at present contending. What is a fair remuneration? How can it be arrived at? The law of supply and demand has an incidence on mining as on other professions, and as regards underground work the manager and mine-captain, who are in daily touch with the labour supplies and the conditions of work, are the most likely persons to form a correct estimate as to what constitutes fair pay. The employer and employee must, however, be free from restrictions, excepting those which regulate the rights of the person and the preservation of life and property. The intervention of Labour Unions and such like are detrimental to the maintenance of a good understanding between the contracting parties. Freedom of contract must be made absolute. Persons not familiar with a mine and unacquainted with the varying conditions of every working place cannot know as much as the contractor, the mine manager, and the mine captain, concerning the equity of the terms and conditions of contracts.

It would be advisable to have long contracts of, say, 3 to 6 months, in order to attract steady men and secure regularity of output. Security should be provided by the contractor, or 25 deducted from each month's payments to ensure the proper performance of the contract. I have not entered into any details as to tendering, etc., but these will readily suggest themselves. Payments must be based upon the work done by the best workmen and not on that done by the inefficient.

The following briefly summarises some of the advantages and disadvantages of the various systems:—

DAY'S PAY.

Disadvantages.—(*a*) A large amount of supervision is required.

(*b*) There is little or no incentive for the miner to do a good day's work.

(*c*) Inefficiency of coloured labour is accentuated where the white labour is inefficient.

(*d*) Carelessness always exists in the use of supplies.

BONUS SYSTEMS.

Disadvantages.—(*a*) A large amount of supervision is needed.

(*b*) Complicated accounts are involved.

(*c*) The conditions of mining work are too varying for any bonus system to be applied with satisfaction.

(*d*) There is always dissatisfaction among the workmen with the results obtained.

CONTRACT SYSTEM OR PIECE WORK.

Advantages.—(*a*) Less supervision is needed.

(*b*) There is greater incentive for both white and coloured labour to do their best work.

(*c*) There is an early start and a late finish of the shifts.

(*d*) Simplicity of accounts.

(*e*) Satisfaction to workmen.

Disadvantages.—(*a*) There is carelessness in following ore bodies, and an unnecessary amount of waste is often mined.

(*b*) To make big fathomage there is an inducement to carry the excavation regardless of changes in the contour of the ore body.

(*c*) There is a difficulty in fixing terms.

In all the systems enumerated the most careful measurement is needed in order to obviate any grievance or misunderstanding. Some mines have an especially bad reputation in this respect.

The writer has heard charges of favouritism levelled at mining officials, but such charges can generally be dismissed: naturally a manager engages contractors upon whom he can rely. When the "man in the street" better understands the great responsibilities resting upon all who are in charge of mining operations he will be less inclined to interfere with the necessarily somewhat autocratic systems of management. Some, who are engaged in directing mining work, may be inclined to look at this question from the technical point of view only, but it is necessary also to insist upon the commercial aspect receiving attention, as it is affected by the varying labour conditions. In discussing the systems for labour payment we must also take into account the maintenance of a regular supply of young men, who will be willing to go through the necessary training, and we have to decide the scale of remuneration which will induce them to follow the occupation of miners.

Several financial experts have called for educational facilities,—Sir George Farrar, Sir Percy Fitzpatrick, Messrs. Lionel Phillips, Chaplin, Fricker, Francke, and others have pointed out in the clearest possible manner the need for reform and development in this connection. All our aims should be directed to the best methods of training a good class of workmen, and, when they are trained, the conditions should be such as to secure for them permanency of employment. The more permanent we make the employment the greater will the efficiency be, and the more attractive will it be to workmen; the wage is also lower, and, as a rule, the better are the economic conditions when constant employment is assured; all these interact on each other. The cleaner we make our mines the healthier they will be, and a greater number of good workers will be attracted to mining.

There are other matters which have an incidence on the contract system of payment, an interesting one being the prices charged for supplies to contractors. Explosives are charged out at prices which show large book profits, but it would appear to be the better plan to charge everything at cost price. Comparisons of costs could then be better made. It may be said that as all contracts are contingent upon the payment of the present general high charges for supplies, no harm is done; but the weakness of this case lies in the fact that the enhanced contract prices for mining, necessitated by these high charges, cause a loss to the mining companies in one direction, balanced only by a "book profit" on the other side.

The President: These few notes of Mr. Austin's are submitted for the purpose of drawing discussion on this important subject of payment and efficiency, and as many of you are well qualified to speak on this matter I hope you will come forward and let us have the benefit of your views. I move a vote of thanks to Mr. Austin for his notes.

THE UTILISATION OF WASTE HEAT IN SLIMES SETTLEMENT.

(*Read at June Meeting, 1907*).

By A. Salkinson (Member).

REPLY TO DISCUSSION.

Mr. A. Salkinson: When writing this paper, it appeared doubtful to me whether it would be possible to obtain an exhaustive discussion on the subject, as until metallurgists have had the opportunity of experimenting with a process, they are naturally averse to hazarding opinions. I am the more grateful to those gentlemen who gave the result of their experience during the criticism and would particularly express my thanks to Mr. Laschinger, who has given me such a welcome opening for reply.

Commencing with the principle that the application of heat accelerates the speed of slimes settlement, I have only heard of two authorities, who have expressed opinions at variance with this statement. The first is the gentleman referred to by Mr. White, and the second, one of our prominent metallurgists who informed me that temperature had no influence on the rate of settlement. It is very easy for everyone to satisfy himself on that point, and in answer I shall only say, "try it."

With regard to Mr. Dowling's remark about the greater decomposition of sulphur compounds at a higher temperature, I quite agree with him that such decomposition would be hastened by heat; but I do not think it is noticeable on a practical scale, at least as far as my own experience serves.

Mr. White's experiments in increasing the amount of lime are interesting, more particularly so as many cyaniders hold the impression, that after reaching a certain point considerably below that mentioned by him, any additional quantity has no appreciable effect on either the rate or density of settlement. I am pleased that he finds no evil effects from the use of mill water of a temperature of 30° C. [86° F.] In our case we have not exceeded 80° F., in fact, the mill water is regulated to be practically of a constant temperature of about 78° F. When he mentions the removal of cooling towers, I do not quite agree with him, suggesting instead that any waste heat necessary might be absorbed by cyanide solution, etc., before the exhaust steam enters the condenser, but that afterwards the usual cooling plant should perform its ordinary work, thus helping the engineering department by the production of a greater vacuum than at present obtainable. In this connection, I may state that recently the sand solutions on the Wit Deep have been heated about 7° F., by passing through a boiler shell on the way to the treatment tanks; the heat is obtained from the cooling water of a condenser passing through the boiler tubes on its way to the cooling towers, thus giving at the same time a little more efficiency to the work of the condenser. The experiment has not been carried on long enough to give definite results, but seems to have had the effect of lowering the assay value of the last drainings to a certain extent.

Mr. Lea deals with the subject in a manner that appeals to the practical man, and again illustrates that an ounce of experience is often worth a ton of theory. His experience was evidently most conclusive, and I am sure that we are all grateful to him for publishing results obtained on a large scale.

Mr. Laschinger quotes some very interesting figures, which I am quite willing to accept up to the point where he applies them to practical deductions; from there his conclusions are erroneous or misleading. He commences by calculating out how much heat is required to raise a ton of water 20° F. This has nothing at all to do with the problem, which I would state in this way. "Given so many hundred or thousand tons of solution, how much heat is required to maintain the temperature of this mass at a given point?" In other words, the heat to be added daily is not that required to raise a quantity from the normal temperature to so many degrees, but only that necessary to compensate for the cooling effect once the total amount has risen to the desired point: for if the temperature of a mass of solution were raised 20° F. in one day, it would still have retained a considerable amount of this heat after the lapse of 24 hours. On a 200 stamp plant, it will take at least a week to obtain this ultimate temperature, and if Mr. Laschinger cares to revise his figures on the above basis, he will find that the final temperature of the solution will be considerably higher and his conclusions wrong. Hence the cost, supposing heat had to be paid for, would be very much less than 3½% on 2 dwt. slimes, and instead of the whole waste heat available being sufficient to raise mill water, sands and slimes solution 17° F., he would find after a little time the temperature very much greater than his calculations show. Having thus started on a wrong basis, he must pardon me if I state that the deductions drawn from his figures are not approximate; they are entirely wrong.

The second part of his criticism is quite on a par with the first half. A good maxim to keep in mind in connection with any proposed alteration is:—Where there is a will, there is always a way. Might I suggest this as an answer to the next paragraph where he talks of the great difficulty of obtaining a reasonable benefit from the waste heat, which the engineers on the Rand are so anxious to get rid of. It is being done on the Wit Deep for one year already, and why can it not be done elsewhere? Instead of the very great expense and the complicated system of control of which he speaks, it is of a comparatively trifling cost and so simple of operation as to be practically automatic. If he or any other member of this Society cares to come to the Wit Deep, I shall be pleased to show him the method, whereby most of the waste heat from a non-condensing engine, and at least a portion of the waste heat from a condensing engine, is being utilised at the present moment for the purpose of heating mill water, sands and slimes solutions. His fears about the mill water getting too hot are quite groundless; our resident engineer, Mr. W. Thomas, informs me that it takes him two minutes per day to regulate that, and as for the cyanide solutions, the hotter they get the better.

Further he states—"it must not be assumed that this so-called waste heat is all wasted because engineers in these days are making strenuous endeavours after efficiency and the prevention of waste of any kind." Why only engineers? For what reason is the metallurgist ruled out of court, who is also striving after efficiency? and if a plan is proposed by which both engineer and metallurgist may gain, why adopt a dog-in-the-manger policy, and deny to the cyanide plant the benefit of waste heat?

Again, I shall be pleased to show him one way, in which waste heat from a non-condensing engine is being utilised solely for the benefit of the engineering department: an installation which was erected as a direct result of the experience gained with my boiler, pump and piping. In fact, I think he must be referring to this identical feature, when he mentions exhaust feed water heaters. Yet there is plenty of waste heat left for far more than all my requirements.

We all know that the subject of heating mill water is not a new one, but I must point out that the reason why we are heating it, is a new one, viz., to obtain a better settlement of slimes; from what I have been able to gather in conversation with mill men, there is no objection to the mill water being a little hotter than normal, as long as the temperature remains constant, and in any case there can be no logical objection to keeping the temperature of the mill water at summer heat during winter months, which was my original intention. I cannot but think that the next paragraph is a little needless, where he says, "The experienced engineer is not looking for troubles and further complications of plant, although he is at all times ready and willing to work out the practical details of any scheme which has for its object the betterment of conditions of working or increased efficiency on the mines in conjunction with either the miner or metallurgist." I think we take all that for granted, but I must say that the way in which I was met by our resident engineer, Mr. W. Thomas, was very different from Mr. Laschinger's attitude. He also was not enthusiastic until he saw how simple the proposed plan was and where his department would benefit; since then, he has been as enthusiastic as any one could desire.

To take the installation of the two boilers fcr the exhaust steam of the winding engines, he informs me that the water condensed far more than pays for the little extra back pressure, which amounts to 3 lb. The cost of the extra pumping I have already given, say, £20 monthly for the two pumps, which is defrayed by the cyanide works and pays for itself many times over. As for the mill water, apart from the benefits derived, it is actually cheaper to run some hot water from the condenser to the return water tanks, as the height to which the water is being pumped is 18 ft. less than the top of the cooling tower. The regulation of the mill water temperature is the only part that requires any supervision, the rest of the heating process is automatic, and there is neither trouble nor complication.

I can only express my regret that he was so unfortunate with the plant for which he was asked to make estimates for the cost of utilising waste heat; I cannot help thinking that it must have been quite an exceptional case. Take, for example, a condensing engine; is there anything impossible in the idea of passing the exhaust steam or even a portion of it through one or several boilers on the way to the condenser? The pump and piping are easy enough, and the strenuous efforts of his engineer will easily overcome any little obstacles.

Assuming that by any chance the solutions should get too hot and the pump be stopped, the engineer would be in no worse case than at present, whilst when solution is passing through the boiler, he obtains a greater vacuum. Another objection that I have often heard raised, is the distance of any available waste heat from the cyanide plant. It may cost more for the greater length of piping necessary, but the cost of pumping liquids over even great distances is not so high, if the proper kind of pump and sufficiently large piping are installed in a proper way.

Mr. Laschinger states that warm water throughout the plant would mean increased water losses by evaporation; I really do not see how that is possible. Under normal conditions so much heat is being wasted, which is dispersed either through the evaporation of water or through conduction. Why more water should be evaporated when this same amount of heat is spread over a larger bulk I cannot understand, "where treatment capacity is ample (as it should be for efficient work)." Now I am quite prepared to grant that there is a large amount of original sin inherent in the ordinary cyanider, because the man who admits that his plant is large enough or good enough has not yet been born. But, assuming that such a plant existed, and it were decided to increase the battery, it would, as he says, be a question of cost whether to erect more tanks for the slimes plant or to instal an arrangement for the utilisation of waste heat. The total amount for the erection of the two boilers, pumps, etc., in connection with the heating of the slimes plant solution on the Wit Deep came to a little under £500. If it had been decided to erect extra tanks, instead of adopting the present system, five additional tanks would have been required at a cost of about £5,000.

His next reference is to the increased amount of dissolved gold caused by raising the temperature. I am pleased to give him the figures he wants, but would ask him whether he would expect the total amount of undissolved gold to remain constant with an increasing charge value?

I would like to repeat my statement, that the benefit of heating slimes solutions and thus getting a better settlement and extraction is proved beyond all doubt on a plant of insufficient capacity. Although our slimes plant is so small, corresponding on a monthly tonnage of 10,000 to 646 cub. ft. per ton treated per 24 hours, including collectors, or 474 cub. ft. per ton treated per 24 hours, taking treatment tanks only, the extraction of gold through the use of heat has greatly improved in spite of a bigger tonnage, as shown by the following figures :—

	Average Monthly Tonnage.	Average Theoretical Extraction.	Average Actual Extraction.
May to Oct., 1906	8,840	75·20%	74·87%
,, 1907	10,309	82·33%	82·17%

N.B.—Gold from slimes dam not included.

Heat, and heat alone has made this difference and if, as I hope we shall by next winter be able to increase our minimum temperature, I can safely predict even better results for the future.

In this connection I would remind you that according to Mr. Laschinger (*April Journal, 1904,*

	Charge Value.	Undissolved Gold.	Percentage of Dissolved Gold.	Remarks.
May to Aug., 1905 ...	1·627	·191	88·26	Ordinary winter temperature.
Nov., 1905, to April, 1906	1·544	·136	91·19	Ordinary summer temperature.
May to Aug., 1906 ...	1·536	·157	89·78	Winter temperature and small amount of heat for 2nd transfer (experimental work).
Nov., 1906, to April, 1907	1·898	·135	92·89	Summer temperature and heat for 2nd transfer.
May to Aug., 1907 ...	2·021	·148	92·68	Winter temperarure and heat for mill water, 2nd transfer and heat for 1st transfer from July.

p. 360) he considers it necessary to have for a 200 stamp mill 1,050 cub. ft. of vat capacity per ton of slimes treated per day, though the average Rand figure is probably nearer 900 cub. ft.

Although Mr. Laschinger questions the simplicity of the process, yet I can assure him that it is quite as simple and inexpensive as I stated, and hope to be able to convince him when he comes to inspect the plant.

In conclusion, I would again express my gratitude to my critics, and more particularly to Mr. Laschinger, who, although he has attacked the process in every way, has, I feel sure, done so in no spirit of unreasoning antagonism, but with a view to stimulating discussion and elucidating points of interest, which might otherwise easily have been missed.

The President: I must congratulate Mr. Salkinson on his reply, and I hope that at some future time he will be able to furnish further figures in connection with this work. I am inclined to think that it would not be a bad idea if we had an excursion to the Wit Deep not only to see this particular process but to inspect the entire surface plant, which is well worth seeing.

Mr. A. Salkinson: Mr. President, I should be very pleased to see you or any other members of the Society there, but it is such a simple thing that you can see it all in about five minutes.

NOTES ON THE ESTIMATION OF CAUSTIC LIME.

(Read at August Meeting, 1907.)

By Edw. H. Croghan (Member).

DISCUSSION.

Mr. E. H. Croghan: Before finally closing the discussion on my paper I wish to state that a copy has been sent to Mr. James Hendrick in Aberdeen, therefore, in view of Mr. Hendrick possibly contributing to the discussion, might I ask that discussion on my paper be kept open?

The President: Yes, certainly.

Mr. E. H. Croghan: There is another matter I wish to mention in consequence of certain queries as to the figures I have obtained for soluble silica. I should therefore like to state the method I employed, so that the process can be discussed, if necessary, and thus place me in a better position to reply to any criticisms. The method was as follows:—

Ten grammes of a sample are evaporated with dilute hydrochloric acid (1 : 2) to dryness on the water bath. The residue is moistened with stronger hydrochloric acid (1 : 1) and the taking to dryness repeated twice. The soluble chlorides obtained are extracted with very dilute hydrochloric acid (1 : 5), the solution heated, filtered off and the insoluble residue well washed. The latter consists of quartz sand, any undecomposed clayey matter, with hydrated silica (if it can possibly exist free after the lime has been burnt) and soluble or combined silica which I have assumed to be taken up by the lime. This insoluble residue is then repeatedly boiled with 5 % sodium carbonate solution, until the alkaline solution, filtered hot, indicates that the extraction is complete. This alkaline silicate liquid is acidulated with hydrochloric acid, and evaporated to dryness, etc., as usual. The insoluble residue which I here obtain is the soluble silica which I have stated in my analyses.

NOTES ON FEEDERS, WITH A DESCRIPTION OF A NEW DRIVING DEVICE.

(*Read at August Meeting, 1907.*)

By D. J. Pepler (Member).

DISCUSSION.

Mr. R. G. Bevington: Mr. Pepler, in the early part of his paper, has given us several points which are, of course, apparent even to a tyro in milling. He mentions one point, however, one which I personally have laid stress upon before this Society, namely, that it is highly necessary that the rock be crushed to a reasonable size in the crusher station: there is no doubt whatever that close attention to this point greatly assists in the output of the mill and also obviates a great deal of unnecessary wear and tear upon a feeder of whatsoever kind, caused by oversized pieces of rock jamming in various places on their journey from the bins to the mortar box. I agree with him that the challenge feeder is about the most suitable form at present known, and if well looked after, there is not a great deal of fault to be found with the three-pawl driving device beyond its multiplicity of parts. With regard to the wear of the pawl and socket, which he mentions, and the slipping of the pawl past the centre, some of you will perhaps remember that in the discussion on, I think, Mr. Roskelley's paper I mentioned a pawl made of wrought iron or steel not straight in form as usually sent out with the challenge feeder but with a set in it in this form ⟨ This pawl starts its work very much farther back from the centre than the straight pawl, and it takes a very long time to wear to such an extent, that it is in danger of passing the centre and falling out. If the pawls get worn a bit short, it is but a small matter for the blacksmith to draw them out $\frac{1}{8}$ in. or so, as may be required. I have seen Mr. Pepler's device at work, and it is certainly very reliable in its action, but I must agree with Mr. Beaver that there are others much less complicated and also very reliable, and which moreover have the advantage of utilising the old friction plates or brake wheels of the challenge feeder. I was interested on reading a paper given before the Institution of Mining and Metallurgy, London, describing certain mills in Western Australia, to see that the author gave a sketch of a driving device (also utilising the old friction disc with the arm cut off) which he claimed to be an excellent one, and which very closely resembles some devices which are in use upon these fields. The sketch of the device is shown in Fig. I.

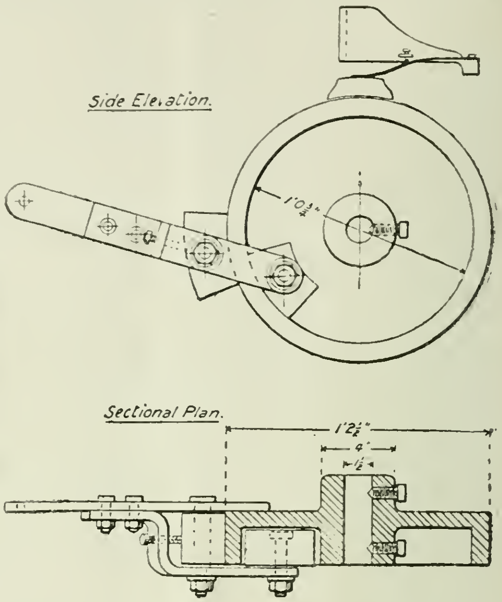


Fig. I.

I may say that I have a very similar device working, which was brought out by one of our Rand mill managers, and find that it works excellently and has required no attention whatever during some six or seven months it has been working, and does not look like wanting any adjustment for a long time yet.

In the discussion upon the above-mentioned paper I see that Mr. W. McDermott, who is one of the highest authorities upon milling matters, mentions a driving device brought out by Mr. Nelson in Western Australia, and which he says is of the simplest possible character, consisting merely of a loose fitting groove in the lever which grips the feed plate (I presume he alludes to what we call the friction disc) by the canting of the lever. No sketch accompanied Mr. McDermott's description, but I have had a driving device made on these lines and find that it works exceedingly well. Anything simpler and less expensive would, I think, be hard to find. I have had one running now for a fortnight and its performance is perfectly satisfactory. I give here a sketch (see Fig. II. on the next page), and should like some of our members to try it.

Mr. G. O. Smart: The drawing given by Mr. Bevington appeals very strongly to me, and I think the device shown is well worth trying. Unfortunately I have no friction discs at the Simmer, as we have not used them for many years, but should I be able to pick up an old disc to suit my feeders I would like to give this drive a

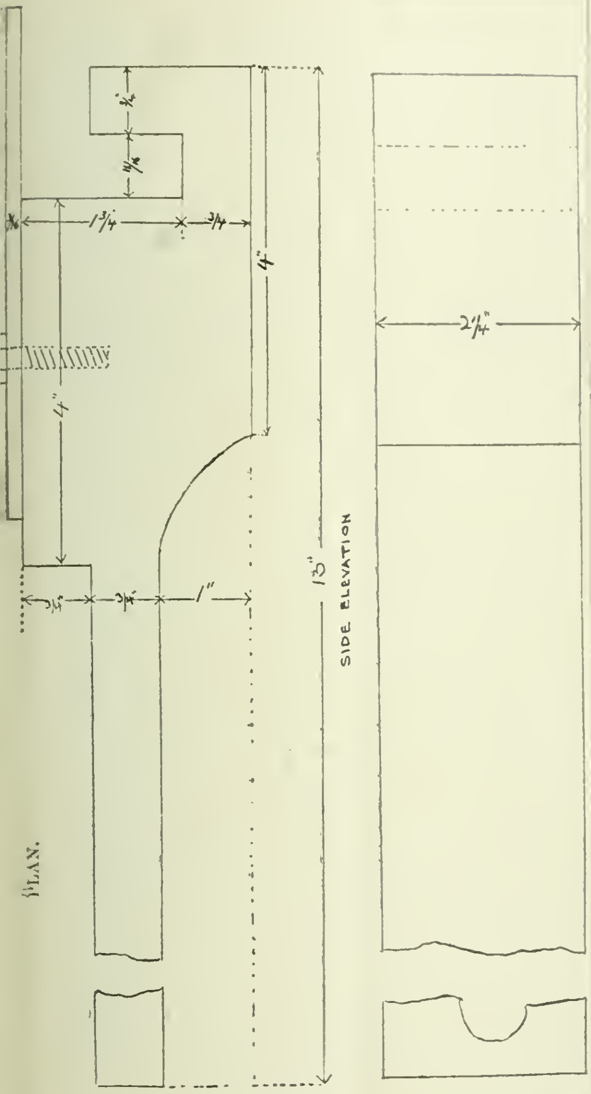


FIG. II.

trial, as it is one of the most simple things we have been shown.

I only propose making a few remarks on that portion of Mr. Pepler's paper dealing with feeders and feeder driving devices, as I think the paper was written principally to bring the author's so-called new driving device, called "Pepler's operative device to challenge feeders" to the notice of this Society. The new driving device is simply another form of the feeder driving gear introduced by me in 1899, to which Mr. Pepler has added some, to my mind, unnecessary complications.

I quite agree with Mr. Pepler that a feeder capable of being adjusted to give a reliable and fine feed is of the utmost importance in a stamp mill. However, like Mr. Beaver, I must take exceptions to the author comparing any modern feeder driving gear with that formerly in use to drive the challenge feeder, as in most up-to-date mills this form of drive has been replaced by one or other of the more modern driving devices now on the market, all of which are more reliable, cost far less for maintenance, and require less attention than the original challenge feeder driving gear. At the same time the original challenge feeder driving device was capable of being adjusted to give a satisfactory feed of ore to the stamps, but required very close attention, and the cost of maintenance was excessive. If Mr. Pepler can save 40 % of stem breakages and increase the stamp duty by 8 %, to say nothing about the cams and cam shaft breakages which he attributes to the original challenge feeder drive, I can only think that very little attention can have been paid to the old device. I thoroughly agree with Mr. Beaver's remarks on the paper.

Mr. R. G. Bevington: In utilising the old friction disc the arm must be cut off and the face of the disc trimmed with a file. That is quite good enough.

Mr. G. O. Smart: In thinking it over, I believe I can pick up some old friction plates on the Simmer and Jack, so I will try the idea and let you have the results.

THE LABORATORY: ITS ECONOMIC VALUE.

(*Read at October Meeting, 1907.*)

By A. McArthur Johnston, M.A., F.C.S. (Member).

DISCUSSION.

Prof. G. H. Stanley: The author draws attention in this important paper to a very real need of the mining industry, and not only of the mining industry but of every other industry using large quantities of stores and of all sorts of material which are supplied to certain specifications, namely, a laboratory where supplies can be regularly and systematically tested or analysed in order to ascertain whether specifications have been complied with or whether the standard set by samples is maintained. The necessity for this constant examination of material is fully appreciated in other industrial centres of the world, even Great Britain being far less backward in this respect than Mr. Johnston would have us believe, and consequently it is a matter for surprise that on these fields, where almost every other branch of the scientific side of the mining industry is so well attended to, so little appears to be systematically done in the direction of testing supplies, the laboratory under Mr. Johnston's charge being, I believe, the only one now maintained by any of the mining groups for this purpose.

The examples given of the work of the laboratory should leave no doubt in the mind of any one as to the value, or necessity even, of the work carried on: since buying on a guaranteed basis is the only way of obtaining good and unfiorm quality, and, therefore, of ensuring that the various operations for which the materials are required shall proceed smoothly and under constant conditions. I have already said that Great Britain is not so backward in this direction. The quotation from *Mines and Minerals* would have applied with equal truth to the old country, though one would undoubtedly not have to make very extended search to discover cases where a very different condition of affairs prevailed. I remember one case where a manufacturer, awakening possibly to the fact that supplies were not up to sample, but with somewhat vague ideas as to the qualifications of a suitable person to attend to this department, requested a college to send him a chemist, who could also fill up his time in helping to keep books and who had a knowledge of bricklaying!

Neither do I share Mr. Johnston's assurance regarding the part played in American advances in metallurgy by the research work of its chemists and metallurgists, important undoubtedly as this has been, and still is. It seems to me, however, that their remarkable progress has been chiefly due to their capacity for making workable or greatly extending other people's ideas, possibly unworkable in the original form, and this by reason of the attention paid to the engineering side of the subject: few essentially new processes have been developed in the States, but processes originating elsewhere have been developed and perfected.

This may seem somewhat beside the point, but it has a bearing on another function of a laboratory which Mr. Johnston touches upon, which is, the testing of new ideas in ore treatment, or of new ores by existing methods: it is needless, perhaps, to say that a laboratory equipped only for testing supplies cannot cope with this class of work, though, of course, very small scale tests can be made which may give extremely useful indications to be followed up in another laboratory suitably equipped for the purpose.

This other laboratory must contain such appliances as crushing and grinding machinery, wet concentrating apparatus such as spitzlutten and vanners, furnaces for roasting and melting, leaching vats, electrolytic vats, filter presses, and so on, so that several tons of the ore under examination may be experimented upon, and this, in my opinion, would constitute a true metallurgical laboratory, not mechanical and metallurgical, or mechanical and chemical, as I think Mr. Johnston should have said; just as a "metallurgist" must be a really good engineer, the term being synonymous with "metallurgical engineer."

Even then, absolute duplication of working conditions is impossible, but I entirely agree with him that such a laboratory would work sufficiently closely to actual conditions to justify, or otherwise, the adoption, or large scale testing, of a suggested process or appliance, with far less risk of financial waste than would be involved by trial in the first case on a large working scale.

Such a laboratory, my colleague, Prof. Yates, and myself hope to be able to get together in the new building of the Transvaal University College, and having this in view, I have dealt with the equipment very briefly, because I hope to have more to say on this point at some future date. But I do not mean to imply that I disagree with the author's suggestion *re* the establishment and operation of such a laboratory by the State, or at least by the co-operation of the mining industry. I think there is plenty of room for both, the college laboratory being primarily intended, of course, for the instruction of students in the principles underlying the practice. As an instance of the instructional use of such a plant I may mention that the students at the Royal School of Mines, in using the small plant there, were taught always to coat the battery plate with silver amalgam at starting, and many such points can be brought before students in such a way that they are not likely to be forgotten in future.

With regard to the examination of coals the amount of volatile matter shown by the method described will vary somewhat with the temperature employed and, therefore, in order to get strictly comparable results it is necessary to duplicate the condition of each determination as closely as possible, though probably the procedure followed will give results close enough for the object in view; but with regard to the ash determination, if the crucible is liable to increase its weight by absorption of lead fumes as also is the ash, I should prefer not to use for incinerations or similar purposes, any muffle which is used for cupellation or scorification.

I may mention here, that when connected with coke works I always used to determine ash in a separate quantity of 5 gm. placed in a small scorifier (not a roasting dish, which has a rougher surface); after the first time or so of use the scorifier method gave practically the same result as by porcelain crucibles and the coal burnt off, much more rapidly owing to such a large surface being exposed to the air; in any case, whether performed in crucibles or scorifiers the coked residue generally took much longer to burn than the original coal, which was rather an important point when 12 to 20 samples per day were being tested. The formula given by Mr. Johnston will only show exactly the percentage of combustible matter extracted if the combustible matter were all of the same kind, *i.e.*, if the volatile hydrocarbons and fixed carbon had the same evaporative factor, but it does show truly the percentage of the total evaporative power which has been developed by burning the coal.

The failure to get a formula for calculating the calorific power from the proximate analysis adds weight to the result of similar failures elsewhere, namely, that it can only be found by actual experiment.

I should like to emphasise Mr. Johnston's remarks with reference to apparatus for measuring high temperatures. Such apparatus is not only a decided gain, but it is in my opinion a necessity. The form I prefer is a thermo-couple provided with a direct reading instrument (a galvanometer), Seger cones are not by any means cheap and can only be used once if they reach a melting temperature, so that such an instrument will soon save its cost, and any number of couples can be used for one reading instrument; the Fèry pyrometer does not, I think, give such accurate readings, and cannot, of course, be used in connection with a recording instrument as thermo-couples can, and this is often of importance in an experimental laboratory.

Mr. Johnston is fortunate in possessing an excellent micrographical outfit, and I am sure we all look forward with great interest to the future paper on this subject, which has now become of such great importance; as to the instance given of its use, wrought iron is at once detected under the microscope, but it must have been a bad wrought iron also to stand for so short a time, and it is in respect to these mysterious causes of failure that the microscope plays such an important part, together with the 100-ton testing machine, in the steel works laboratory, the maker usually being, for the sake of his firm's reputation, as anxious to supply as the user is to buy only material which can be depended upon, he cannot afford to risk the rejection of his material as being not up to specification. Wrought iron is not made on anything like so large a scale as steel, and largely by comparatively small and conservative works, where the testing is not carried out, or, at any rate, is less rigid, so that the quality naturally tends to be variable.

In conclusion, I congratulate the author on giving us a paper which provides so much material for thought, and which cannot fail to be productive of a most interesting and instructive discussion.

Mr. J. H. Harris: I should like to confirm Mr. McArthur Johnston's statement with regard to setting new copper plates with silver amalgam. We adopted this method at the Jumpers Deep in March, 1906, for setting four out of eight plates of our two tube mills. The silver employed was in the form of 771 three penny pieces, weighing about 30 oz. troy; these were rolled out thin and dropped into about a quarter of a bottle of hot mercury (just below vapourising point) to form a paste. This amalgam was then rubbed in in the usual way, and gave perfect satisfaction. The plates have always been in as excellent order as the other four, which were set with gold amalgam.

Mr. W. A. Caldecott (contributed): Considering that several million pounds are spent here every year on the purchase of mining supplies, the subject of Mr. McA. Johnston's important paper, with which he is so well qualified to deal, is worthy of the serious consideration of every one connected with the mining industry. The systematic testing of materials has perhaps reached its highest state of development in the United States, and its foremost exponent may be taken as the Pennsylvania Railroad Company, a colossal corporation operating or controlling over 23,000 miles of track, and with gross earnings amounting to some £60,000,000 yearly.

I was recently privileged to look into the system employed at Altoona by the testing department of this company, whose chief chemist, Dr. C. B. Dudley, is likewise president of the American Society for Testing Materials. Some 25 men are employed on the chemical side alone of the above testing department, and rather more than that number are engaged on mechanical tests and photo-micrographic work.

Detailed specifications, which represent in some cases the labour of years, and in framing which, not only the actual user but the seller is invited to co-operate, are employed in the purchase of all material possible, combined with a complete system of inspection and sampling. These specifications for lubricants, castings, hose, illuminants, paint, soap, cement, and scores of other articles provide that rejected shipments will be returned at the cost of the shipper, but since sellers have in course of time realised that ingenious sophistications will be detected, and that nothing but a good article will pass, the percentage of rejections due to supplies in bulk not equalling samples is exceedingly low.

From time to time, with increased experience, specifications are withdrawn and replaced by improved forms, and in many cases specifications are accompanied by a detailed description of the method of analysis to be employed. The advantages of such a system to one of the best organised corporations in the world is shown by its development, since its inception a quarter of a century ago. The first-class manufacturer likewise legitimately profits thereby over his inferior competitor, since bias and the chance influence of a more persuasive advocate are reduced to a minimum.

At the Schenectady works of the well-known General Electric Company, which I was also courteously permitted to visit, a similar system is in vogue, and the head of the large testing department of this company likewise determines what materials shall be employed for each purpose in the manufacture of the company's electrical goods.

At the Woolwich Arsenal, near London, great attention is paid to the regular photo-micrographic examination of the steel used in making big guns, and here as elsewhere this system serves as a means of detecting inferior quality before use, as well as serving to elucidate in great measure the cause of weakness or failure in service.

EXPERIMENTS IN FIRE ASSAYING AT THE REDJANG LEBONG MINE, SUMATRA.

(*Read at September Meeting, 1907.*)

By G. B. Hogenraad (Associate).

DISCUSSION.

Mr. L. J. Wilmoth: This paper is of great interest to us Rand assayers, as it gives us a glimpse of the assay practice on a mine in a far off field. At a first glance of the analysis submitted it would seem that the ore was complicated, but on a closer inspection it is in reality very simple indeed. The metallic oxides mentioned are present in such small quantities as not to have any prejudicial effect on the accuracy, with exception perhaps of the manganese.

The author is inclined to blame the borax as being the cause of the excessive losses during fusion. No doubt, the upholders of the idea that borax is an "effete superstition" will agree with him, but to me the borax is not the cause of the losses mentioned. If the manganese mentioned in the analyses be present as a dioxide, we have at once a factor inclined to give a heavy slag loss, particularly of the silver. The proportion present is small, but, no doubt, would exert a certain effect in that direction. Considering the effect of the manganese and the high value of the silver contents of the samples, the question arises, as to whether the Reduction Works samples should not be assayed by scorification. Still with an ore of this composition there should be no difficulty in obtaining a satisfactory fusion in a crucible. In his first charge, he relies entirely on his soda as a fluxing agent, as he reduces practically all his litharge to metallic lead. I think that had he altered his proportion of soda and litharge to, say, 1 of soda and 2 of litharge and so made litharge his predominating flux he would have had no trouble from the start. The curious fact stated by him, that the elimination of borax from the flux decreases the silver losses, is surprising, and I regret that I have not any material of the same values to experiment on. It seems to me that the silver losses were due in the first charge, not to the borax in particular but to the general unsuitability of the flux for the charge operated upon.

His advocacy of the abolition of borax from the crucible assay once again raises a point of great interest to us Rand assayers. Several assayers are against its use, although the great majority of us still adhere to it. I have at different times tried a few experiments to prove for my own satisfaction what was the proper course to adopt. I have tried various proportions of soda and litharge, and various proportions of flux to ore, and I have found that a satisfactory fusion can only be obtained with a greatly predominating proportion of litharge present, and, though this may give a fairly satisfactory fusion, it certainly does not give higher results, and the assay is very expensive. With a bulk flux of 1 of soda and 1 of litharge and a proportion of 4 of flux to 1 of ore as advocated by some, the slag is thick and pasty and requires a very high fusion heat to obtain a slag that will pour at all. This heat question is a serious one—in a large office putting through several thousand assays per month. For instance, with 1 A.T. of ore mixed with 4 A.T. flux consisting of 1 part litharge, 1 part soda and $\frac{1}{4}$ part borax I can comfortably fuse 20° F. crucibles in a fire and have them poured within 40 minutes of placing them in the fire, the fuel used being the ordinary Natal coke. Now I find that by using the soda and litharge flux with a proportion of 4 of flux to 1 of ore I have to leave the crucibles nearly half-an-hour longer, and even then there is action still going on in the crucibles. This extra 30 minutes is a serious question, as the Natal coke is inclined to burn away quickly, and has a tendency to fail the assayer at a very critical time if the non-borax flux be used. If I were to adopt—granting the results were as good, which I find they are not—the non-borax flux, I would have to use English coke in my office instead of the Natal coke, which would mean a fair amount of money, when the

Natal coke is between £2 and £3 per ton cheaper than the English.

There is no doubt that a well fused charge of banket ore using borax in the flux yields higher results than with a non-borax flux, due, I have no doubt, to the extra fluidity of the charge.

Mr. Hogenraad mentions that he reduces 31·4 gm. of lead per 1 gm. of charcoal. The amount of lead seems high, as 30 gm. is high for the usual assay charcoal. Perhaps he uses a charcoal fire for his fusions or has forgotten the presence of a small amount of sulphur in his charge.

The cracking of crucibles after a rainy night seems strange, but I fail to see the connection of the large amount of litharge and a cracked crucible unless they were both left out in the rain by accident.

The third charge, which yields such a large button that it needs scorification, can hardly be termed satisfactory, though it might be made so by reducing the amount of charcoal added. The necessity for scorification is a very serious drawback to the success of any charge. That the scorification should have a tendency to yield higher results seems to point to a source of error due to some impurity in the lead button, and this, no doubt, is the selenium originally in the ore. An assay of his cupels would be exceedingly interesting.

In conclusion, I would like to suggest that if he were to adopt a charge similar to those used on the Rand with the exception, perhaps, of a higher proportion of litharge, on account of the selenium present in his ores, he would find his troubles considerably lessened.

Mr. H. A. White: I should like to point out that if Mr. Wilmott will reduce his proportion of flux from 4 to 1 down to 3 to 1 he will find that that extra 30 minutes will also be eliminated.

NOTES ON SMALL STOPE DRILLS.

(Read at October Meeting, 1907.)

By E. M. Weston (Associate).

DISCUSSION.

Mr. E. M. Weston: Since writing the above paper I have noticed two mistakes, which arose from relying on my memory only. The first is that the present type of large Leyner machine does not use an anvil block but strikes the blow direct on to the specially hardened shank of the boring tool. The second is that the stroke of the Gordon drill is from 8 to 8½ in., not 4 in. as stated in my paper, and the hammer is 1¼ in. diameter and is under 2 lb. in weight.

Mr. R. H. Anderson: I have been much interested in Mr. Weston's paper dealing with the various kinds of rock drills and their possible development. Recently I have tried several well known makes of rock drills under precisely the same conditions, and in my opinion there is little difference in speed of drilling between any of them.

Mr. Weston hits the nail on the head, however, when he says that the fastest boring machine is the most economical. At the same time he only mentions drill steel in a casual way, and does not appear to appreciate the loss in drilling due to imperfect design and dressing of the drill steel. Two years ago Mr. Cameron, of the May Consolidated G. M. Co., suggested to me that the only way of getting over the present troubles was some scheme of detachable bits, and from that time to the present I have been engaged with Messrs. Thos. Firth & Sons, Sheffield, in perfecting the manufacture of the detachable bit that I am at present introducing on these fields. Up to the present I have demonstrated the capabilities of these bits on the following mines:—May Consolidated G. M. Co.; Village Deep G. M. Co.; Wolhuter G. M. Co.; Meyer and Charlton G. M. Co.; Simmer and Jack G. M. Co.; New Primrose G. M. Co.

Using a 3¼ in. Ingersoll on the May Consolidated worked by two Kafirs, we drilled 13 holes in 7 hours 40 minutes, averaging 5 ft. 6 in. deep, or a total of 71½ ft.; 52 bits wers blunted in doing this work. The contractor had two other machines at work in the same stope which drilled 5 holes each in the shift. Had these machines been fitted with my bits a total of 39 holes would have been drilled with ease or 214½ ft., and 156 bits would have been blunted, against 81 ft. as drilled under the present conditions.

Simply regarding my drill as a means of increasing the speed of drilling the increased efficiency is due to the following factors:—

(*a*) No waiting for drill steel.

(*b*) Never using a bit twice.

(*c*) Drills always being of standard lengths.

(*d*) The gauge or diameters being accurate.

(*e*) All bits being interchangeable (that is the starting bit could be fixed to the finishing shank or *vice versa*).

(*f*) The cutting edges being arranged that they always form a round hole.

(*g*) The end faces of the cutting edges form segments of the same circle and are of such a size that they wear evenly and without friction.

(*h*) Ample clearance is allowed.

(*i*) The permanent drill shanks are made much heavier than the present steel, and are capable of properly delivering the blow without loss of energy due to bending.

(*j*) The combination of an improved arrangement of cutting edges with the increased strength of shank enable holes to be collared at angles which the present steel cannot attempt.

(*k*) Due to the precise gauging of the bits holes can be drilled, starting with a less diameter of bit and finishing larger than with the present steel.

If any one present would care to see my detachable bit at work I would be glad to make arrangements.

The President: I am very glad that Mr. Anderson has brought up this matter of detachable bits, and I regret that he has not with him a few specimens to show us, because there are many here who have not much time to run into town, and it would have been a good opportunity for them to see exactly what the device was like. If Mr. Anderson does not mind I would like a little more information on this subject. In the first place, what is the cost of these bits, and how much time is lost in changing them ?

Mr. R. H. Anderson: The cost of the detachable bits is simply a question of quantity. If 10,000 bits were required each day the cost would be quite reasonable :—The time taken to replace a blunt bit varies with the length of the drill, the starter can be replaced in 40 seconds, while the 6 ft. or 7 ft. shank takes 1 minute 30 seconds. The changing of the bits, however, does not interfere with the work of drilling, the starter being changed while the second drill is at work, and so on.

The President: I am afraid I am very inquisitive, but I want further information. You have suggested a drill with a detachable cutting edge. Your idea I suppose is that you renew or retain the sharpness far longer than the ordinary cutting drill. Which do you consider the more important, the stronger shank or the sharper bit ?

Mr. R. H. Anderson: The effect of the strong shank is to deliver a heavier blow as the minimum of energy is lost in the give of the shank. This condition naturally causes more wear on the cutting edges. I lay more importance on the arrangement of the cutting edges and especially on the fact of the bit forming a round hole and having correct guiding faces which work without friction in that hole and keep the bit concentric with the axis of the piston rod than on the degree of sharpness retained by the chisel edges. In fact, the drill that has been binding on the side of the hole keeps its edge better than one which has worked freely. I drill much faster than the ordinary drill steel and consequently blunt my edges more.

The President: Perhaps at the next meeting you may be able to exhibit some of the bits ?

Mr. R. H. Anderson: Yes.

Mr. H. H. Johnson: I have had the pleasure of seeing the Anderson detachable bit at work, and there is no doubt as to the saving to be made with it. Mr. Anderson seems to have hit the right idea. Whether his scheme will be a commercial success depends upon what price is charged for these bits. But there is no doubt whatever there is a great saving, and a great increase in the efficiency with the use of a stouter drill. There is also a saving in labour. If these bits come into general use it will obviate men using blunt jumpers simply because the sharp ones are too far away.

The President: Cannot you give us a little contribution on stope drills, Mr. Johnson ?

Mr. H. H. Johnson: I hope to do so at a later meeting.

Mr. C. H. Wilkins: I would like to point out a serious error in Mr. Weston's paper. In the ordinary type of piston rock drill he calculates the volume of air used as being the volume swept through by the piston for the full stroke. He is probably not aware that air is used expansively in most piston drills. I know of one drill in particular which cuts off at half on the forward and return strokes.

Mr. E. Lawrie: During the last two days I have had the pleasure of being at Mr. Anderson's test at the May Consolidated, and I would like to explain a few points he has not made quite clear. You asked if any time was wasted in detaching the bits. He has two sets of shanks down below and whilst he is using the first the next is made ready with fresh bits. When the first shank has got as far as it can go the second shank is put in, and then a third with a fresh bit. Then he starts another hole with a fresh shank so that no time is wasted. Even with only one set of shanks we can drill without wasting any time, but, of course, it is advisable to have a second set in case of accidents. Then with regard to the difference between a bit with fresh cutting edges and blunt cutting edges, we found that with a blunt bit on a short shank the latter drilled, as near as could be judged, as rapidly as if the bit was a new one, thus substantiating Mr. Anderson's statement that rapidity of drilling is due to the strength of the shank.

The proceedings then closed.

Contributions and Correspondence.

WEAR OF STEEL IN HAND DRILLS.

About two months ago a remark by Mr. Morton, of the Vogelstruis Estate Co., on the considerable wear of steel at the head or hammer end of hand drills, led the writer to have the following test made on the Windsor mine, and the results may be of interest to some of the mining members of the Society.

Fifty drills were cut from the ordinary steel in use on the mine, $\frac{3}{4}''$ octagon of good quality: the heads of 25 were hardened in cold water at the dullest observable red heat, the other 25 were not hardened. The whole 50 were marked, mixed with the drills in use on the mine, and measured after a month's use under ordinary conditions.

Hardened Heads.—

Wear at chisel end, max. $\frac{3}{4}''$, min. $\frac{1}{4}''$, average $\frac{1}{2}''$.
Wear at head end, nil, but a few drills were chipped.

Unhardened Heads.—

Wear at chisel end, max. $\frac{3}{4}''$, min. $\frac{1}{4}''$, average $\frac{1}{2}''$.
Wear at head end, max. $2\frac{1}{2}''$, min. $1''$, average $1\frac{5}{8}''$

It may be added that a single shift only is worked in the mine, and as the ground is easily drilled the chisel ends rarely require more than a re-dressing. It may also be added that the drill boys look on the hardened headed drills with considerable disfavour.

S. H. Ford.

Luipaardsvlei,
October 23, 1907.

THE ELIMINATION OF GOLD BEARING SOLUTION FROM SANDS.

By W. A. Caldecott and A. McA. Johnston.

The practicability of washing sands residues, containing gold in solution, by dropping these through a column of water was experimented on some few months ago by us. Given sufficient depth of water or precipitated cyanide solution, there can be no doubt that a very thorough elimination of the gold solution from the sands can be effected. The following data were obtained, *inter alia*, in these tests:—

The apparatus consisted initially of a top vessel, the container, in which the sands with gold bearing solution were placed, a tube connecting this with an enclosed vessel, the receiver, containing water and into which the sand fell.

1. The upper funnel contained to begin with 1,500 gm. of sands, mixed with 1,000 c.c. gold bearing solution, valued at 3·50 dwt. per ton. The bottom receiver held 1,350 c.c. of water and the $\frac{3}{8}$ in. tube connecting the same is credited with 270 c.c. of water. At the end of $2\frac{1}{2}$ hours all the sand had dropped into the bottom vessel and the solutions were found to assay as follows:

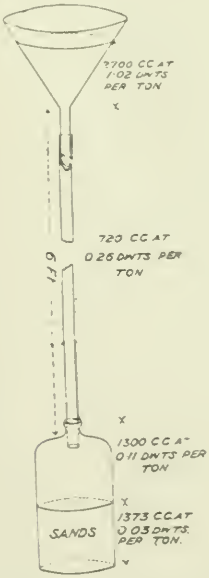
1,620 c.c. in the top vessel = 1·78 dwt. per ton
270 c.c. in the $\frac{3}{8}$ in. tube = 0·90 dwt. per ton
280 c.c. in the bottom vessel above sand ... ... = 0·77 dwt. per ton
450 c.c. in contact with the sands ... ... = 0·34 dwt. per ton

2. Using a 6 ft. length of tubing to connect the two vessels and a larger receiving vessel, the solution in contact with the sands was brought down even lower.

The tubing measured $\frac{3}{4}$ in. inside, and at the end of $2\frac{1}{2}$ hours when all the sands, 3,062 gm., had been transferred, the solutions assayed as follows:—

2,700 c.c. in the top vessel = 1·02 dwt. per ton
720 c.c. in the $\frac{3}{4}$ in. tube = 0·26 dwt. per ton
1,300 c.c. in the bottom vessel above sands ... = 0·11 dwt. per ton
1,373 c.c. in contact with the sands ... ... = 0·03 dwt. per ton

3. Dropping the sand through a screen fitted into a large cylindrical vessel showed fairly satisfactory results except that the solution other than that in contact with the sands mixed thoroughly. The supernatant solution assayed 1·24 dwt. per ton, whilst that in contact with the sand after washing assayed 0·27 dwt.



Further experiments in this direction were carried out, but none gave as good results as in Nos. 1 and 2. The objectionable feature was the failure, as in experiment 3, to obtain layers of gold bearing solution of different values which could have been decanted off to separate storages for treatment. The solution in contact with the sand was generally of comparatively low value. The working of this method of washing sands was demonstrated to members at the meeting, and it was also mentioned there that provisional patent rights had been taken out.

Obituary.

The following deaths are recorded with regret: Mr. HENRY ASHER TILGHMAN (Member), formerly Assistant Consulting Engineer to the Chartered Company, Bulawayo, whose death is reported from California. Mr. Tilghman was elected a member in November, 1904.

Mr. JULES PERRIN (Associate), Assayer of the Mint at Pretoria, under the Republican Government. Mr. Perrin joined the Society in June, 1898.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

THE DIRECT MEASUREMENT OF OSMOTIC PRESSURE.—"The ordinary direct method of measuring osmotic pressures is to obtain equilibrium on the two sides of the semi-permeable membrane by means of the pressure of a head of liquid. The method devised by the author and Mr. E. G. J. Hartley substitutes mechanical pressure, which is put straight on to the solution, and equilibrium thus obtained. Reversing the usual arrangement, the solution is put outside the semi-permeable tube, in an outer containing tube of gun-metal, and the solvent inside, suitable mechanical arrangements being provided for rendering the joints leak-tight. The pressure is obtained from a plunger worked with a pile of weights and delivered through a mercury U-tube to the solution outside the septum. Equilibrium is obtained or calculated by watching through a telescope the motion of the solvent in a gauge. The measurements may be made at any desired temperature. Corrections are applied for the imperfect semi-permeability of the membrane by analysing the solvent to see if any solution has passed into it.[*]

A vapour pressure method for measuring osmotic pressure was also described.[†] This is a modification of the Ostwald and Walker dynamical method. Air is sucked through a train of glass vessels containing first sulphuric acid, to dry it, then through the solution from which it gets saturated up to the vapour pressure of the latter, then the solvent, of which it here takes up more, and finally sulphuric acid again. There are two trains of vessels arranged to oscillate about a central axis, so that the air does not actually bubble through the liquids, which is unsatisfactory, but merely passes over them, small platinum tubes being placed in the tubes to increase the wetted surface. The whole apparatus can be placed in a bath and worked at any desired temperature. Arrhenius' relation modified for concentrated solutions was used for calculating the osmotic pressures from the lowered vapour pressures, and very concordant results were shown to have been obtained between measurements made by the direct-pressure and the vapour-pressure methods. The author is of opinion that the latter method will chiefly have to be relied upon to furnish accurate data for the further study of osmotics.

Both of these methods are described in full in the papers referred to below."—THE EARL OF BERKELEY. —Transactions of the Faraday Society, July, 1907, p. 12. (H. A. W.)

THE LITHARGE METHOD.—"The efficacy of the litharge method depends upon the following facts: 1st, that litharge when melted with sulphides and compounds of the base metals oxidises both the metals and the acid elements with evolution of the volatile oxides (as SO_2, As_2O_3, etc.); 2nd, that molten litharge will dissolve and form easily fusible mixtures with the oxides of metals that are in themselves difficultly fusible; 3rd, that a portion of the litharge is reduced during this oxidation to metallic lead which collects the unoxidised gold and silver.

The slag is very fluid at a moderate heat, and on this account there is a cleaner separation of slag and lead than is the case with other crucible methods. The lead button breaks out soft, clean and bright and need not be hammered at all.

Chemically the method is nearly perfect, practically all ores and metallurgical products being both completely decomposed and fluxed. It reduces cupellation losses by oxidation and subsequent removal by volatilisation or solution in the slag, of those elements which when present in the lead button during cupellation increase cupel-absorption losses, as sulphur, arsenic, antimony, tellurium, copper, zinc, iron, etc. This without previous roasting or the tedious and uncertain recovery of values from matte or speiss sometimes formed in the fusion by the old standard methods. Gold, no matter what its condition in the material treated, is practically completely recovered. Silver requires special precautions to prevent loss of a small percentage. It has been found that a portion of the silver content enters the slag, and is not recovered unless a second precipitation of lead is made after the first lead precipitated has settled. This is accomplished by placing one or more crucibles of coal in the closed muffle with the fusions, and generating gas enough to reduce about 2 gm. of lead after all other action has ceased. The additional recovery of silver (about 2% of silver content) may be due merely to the mechanical collection of suspended metal, but if this were the case, gold should show a similar result. This is not the case, there being no loss of gold as with silver, and the writer believes the loss to be due to oxidation of a portion of the silver.

The separation and slagging of copper is not complete, though more so than with other methods involving only the one separation. It will be noticed that the litharge method depends upon the same principles as scorification and cupellation, as regards elimination of base metals, *i.e.*, the oxidation of the latter and their solution in molten litharge.

From a practical standpoint the litharge method presents only two disadvantages. The cost of fluxes is somewhat higher than for most other methods, but where accuracy is desired this is of slight consequence. The most serious objection to the scheme is the danger to the health of the operator through absorption of lead into the system. Where such large amounts of lead oxides are handled and the air is fouled by the fumes from the fusions, the danger of lead poisoning, always to be guarded against in an assay laboratory, is greatly increased. This subject of lead poisoning among assayers is deserving of more attention than has been given it, and the ventilation of laboratories should be carefully considered, where the litharge method is in use.

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[*] See also Trans. Roy. Soc., vol. 106, 1906, p. 481.

[†] See also Proc. Roy. Soc., vol. 77, 1906, p. 156.

In the practical operation of the method only three stock fluxes are needed to treat any material, one for silicious ores, one for basic ores and one for mattes and copper bullion. These may be made up in quantity by the following formulæ.

For silicious ores (over 50 % SiO_2) :—

24 lb. litharge.
2 lb. sodium carbonate (Na_2CO_3).
2 lb. potassium carbonate (K_2CO_3).

For basic ores :—

24 lb. litharge.
2 lb. sodium carbonate.
2 lb. potassium carbonate.
3 lb. silica.

For mattes, etc. :—

32 lb. litharge.
1 lb. sodium carbonate.
1 lb. potassium carbonate.
2 lb. silica.

It has been found that the admixture of a small amount of alkaline carbonate and silica improves the slag, reduces corrosion of the crucible and makes a cleaner, more coherent lead button, than does the use of litharge alone ; hence they are always used. The silicious ore flux differs from the basic only in the absence of silica, which is furnished by the ore itself in such a case. In the treatment of ores, four or five assay tons of stock flux are placed in a 20 gm. crucible and one-half assay ton of ore added, together with reducing agent (sulphur) or oxidising agent (nitre) as indicated.

After mixing, the charge is covered with a layer of salt, and placed in the muffle at a low red heat, which is gradually raised for about 40 minutes, when the crucibles may be poured.

In the case of material carrying high silver values, to obtain reliable results it is necessary to observe the precautions previously mentioned, *i.e.*, close the muffle tightly, and maintain a reducing atmosphere during the fusion by placing one or more crucibles of bituminous coal in the muffle with the charge. With a gas or gasolene furnace, there is usually a reducing atmosphere in the melting furnace, in which case no coal would be required."—B. M. SNYDER.—*Western Chemist and Metallurgist*, July, 1907, p. 125. (H. A. W.)

VANNING ASSAY OF TIN ORE.—"In dealing with Cornish methods generally, a recent contribution to your correspondence columns specially singled the above out for criticism, alleging that only some 75 per cent. of the tin actually present could be obtained by this method. As this opinion is not uncommonly held in some circles it may be of interest if I furnish some extracts from a paper recently contributed to the *Camborne School of Mines Magazine*, by Ernest Terrell, then assayer at Clitters United, and now manager of the Stormsdown and Owlacombe tin and arsenic mines.

Made up samples were used, the matrix materials having been previously analysed for tin and found pure in that respect.

Sample No. 1, all through 30, none through 120 mesh : Silica, 40 per cent. ; iron oxides, 25 per cent. ; mispickel, 15 per cent. ; iron pyrites, 10 per cent. ; copper pyrites, 8 per cent. ; tin oxide, 2 per cent. Tin oxide recovered by vanning, 1·92 per cent., equivalent to 96 per cent. recovery.

Sample No. 2, all through 150 mesh : Silica, 50 per cent. ; mispickel, 10 per cent. ; iron oxides, 35 per cent. ; copper pyrites, 3 per cent. ; tin oxide, 2 per cent. Tin oxide recovered by vanning, 1·887 per cent., equivalent to 94·35 per cent. recovery.

Sample No. 3, all sulphides roasted before vanning : Iron pyrites, 60 per cent : copper pyrites, 15 per cent. ; mispickel, 23 per cent. ; tin oxide, 2 per cent. Tin oxide recovered by vanning, 1·90 per cent., equivalent to 95 per cent. recovery.

Tests on natural samples were also made with the following results, the chemical assay employed as a check being the hydrogen reduction method :—

TIN OXIDE.

	By Chemical Analysis.	By Vanning.	Recovery.
No. 1 ...	4·26	4·01	94·1
No. 2 ...	0·64	0·60	93·7
No. 3 ...	0·22	0·20	90·9

While the above figures, and others with which I will not trouble you, as they are readily accessible to anyone interested in the question, speak highly for the assay by vanning, there is no doubt that a really rapid, accurate, and easy chemical method is greatly needed. It is not generally known that native tin oxide can be reduced by acting upon it with hydrochloric acid in the presence of zinc (or of magnesium, aluminium, or other high potential metals) ; but unfortunately the reduction is never complete, some oxide always obstinately resisting attack. I have found that reduction is assisted by allowing the hydrogen evolved to accumulate under pressure, and would suggest further experiment along this line by anyone setting out to solve the problem."—R. T. HANCOCK.—*Engineering and Mining Journal*, July 6, 1907, p. 31. (A. McA. J.)

A NEW METHOD OF DETERMINING AMMONIA.—"The method depends on the fact that, in the presence of a large excess of formaldehyde, ammonium salts form hexamethylenetetramine, liberating the acid quantitatively. *Determination of ammonia in a neutral salt.*—The neutral solution is diluted to 100 c.c. with distilled water, free from carbonic acid, and a few drops of phenolphthalein indicator, and a large excess of 20 per cent. formaldehyde solution (previously neutralised) are added. The free acid is then titrated with *N*/10 sodium hydroxide solution. *Determination of ammonia in presence of free acid.*—Before adding the formaldehyde solution, the liquid must be neutralised, but the presence of ammonium salts renders the end-reaction with phenolphthalein wanting in sharpness, and a suitable indicator must therefore be chosen."—A. RONCHESE, J. Pharm. Chim., 1907, 25, 611–617.—*Journal of the Society of Chemical Industry*, July 31, 1907, p. 841. (A. W.)

THE QUALITATIVE SEPARATION AND DETECTION OF FERRO-, FERRI- AND SULPHO-CYANIDES.—"The solution to be tested, preferably dilute and about 5 c.c. to 10 c.c. in volume, is acidified faintly with acetic acid or hydrochloric acid and treated with a soluble thorium salt to complete precipitation. To the liquid with suspended thorium ferrocyanide some finely shredded asbestos is added, the whole is agitated and thrown on a filter, and the precipitate washed with a little water.

The presence of the ferrocyanide may be confirmed by decomposing the washed precipitate with strong sodium hydroxide on the filter, acidifying the clear filtrate with hydrochloric acid, and testing with ferric chloride.

The filtrate from the thorium ferrocyanide is treated with a soluble cadmium salt to complete precipitation of the cadmium ferricyanide, which after the addition of asbestos is filtered and washed as previously described. The presence of the ferricyanide may be confirmed by treating the cadmium ferricyanide with sodium or potassium hydroxide, acidifying the filtered solution, and testing with a ferrous salt.

The filtrate from the cadmium ferricyanide is acidified with hydrochloric acid and treated with ferric chloride, which gives the red ferric sulphocyanide.

The method is especially recommended for the detection of small amounts of ferrocyanides and ferricyanides in the presence of each other. The following table gives some results obtained :—

Tests for $K_4FeC_6N_6$ *only.*

$K_4FeC_6N_6$ present. gm.	$K_3FeC_6N_6$ present. gm.	KSCN present. gm.	Indication.
0·0010	0·1	0·1	Distinct.
0·0005	0·1	0·1	
0·0002	0·1	0·1	
0·0001	0·1	0·1	

Tests for $K_3FeC_6N_6$ *only.*

(These tests were made without waiting to wash the ferrocyanide precipitate thoroughly.)

0·1	0·0010	0·1	Distinct.
0·1	0·0005	0·1	Fairly distinct.
0·1	0·0002	0·1	Faint.
0·1	0·0001	0·1	Very faint.

Tests for KSCN *only.*

0·1	0·1	0·0010	Distinct.
0·1	0·1	0·0005	
0·1	0·1	0·0002	
0·1	0·1	0·0001	

Tests for $K_4FeC_6N_6$, $K_3FeC_6N_6$, *and* KSCN.

0·0100	0·0100	0·0100	Good tests for $K_4FeC_6N_6$ $K_3FeC_6N_6$ and KSCN.
0·0050	0·0050	0·0050	
0·0010	0·0010	0·0010	

Tests of Mixtures Unknown to Analyst.

0·0010	0·0010	—	Found : $K_4FeC_6N_6$ and $K_3FeC_6N_6$.
0·0010	—	0·0010	Found : $K_4FeC_6N_6$ and KSCN.
0·0010	0·0010	0·0010	Found : $K_4FeC_6N_6$, $K_3FeC_6N_6$, and KSCN."

—P. E. BROWNING and H. E. PALMER, *American Journal of Science*, [4], xxiii., No. 138.— *The Chemical News*, vol. 96, July 5, 1907, p. 7. (A. W.)

SILICA WARE.—" The ability to fuse silica into a pure transparent mass has previously been announced in the scientific journals.

Fused silica does not crack on subjection to the most violent and sudden changes of temperature. It is unattacked by the volatile acids, with the exception of hydrofluoric acid. It has a melting point approximately equal to that of platinum. It is harder than ordinary glass. Above 1,000° C. it is permeable to hydrogen and certain other gases. Its coefficient of expansion is 0·00000059 per ° C. (about $\frac{1}{17}$ of that of platinum). Its expansion, up to 1,000° C., is regular : above 1,200° C. it contracts. As far as is at present known, it shows no tendency to devitrification. Its density is 2·2 (approximately)." —*Engineering and Mining Journal*, July 13, 1907, p. 77. (G.H.S.)

RHODESIAN MINERALS—CHROMITE.—The chemical analysis yielded Cr_2O_3, 46·36%, Al_2O_3, 13·18%. Ferrous and ferric oxides (calculated as FeO), 18·66%, NiO + CoO, 0·17% MgO, 13·64%, SiO_2, 4·58%, H_2O, 2·72%. The presence of Pt in traces was also ascertained.

Antimony Ore.—On analysis the following were obtained, Sb, 65·67%, Fe_2O_3, 0·3%. Residue insoluble in acids (chiefly quartz) 7·95%. No gold or silver was found. The ore proved to be stibnite of good quality.—*Bulletin* Imperial Institute, vol. v., 2, 1907, pp. 137—8. (J. A. W.)

BICHLORIDE OF MERCURY IN NITRO-GLYCERINE EXPLOSIVES.—The late Dr. Dupré, chemical adviser to the British Government under the Explosives Act, reported that mercuric chloride has a remarkable effect in prolonging or masking the heat test, even when present in extremely small quantities. The effect is due to the metallic mercury derived from the chloride by reducing agents present, such as metallic iron derived from the manufacturing machinery : and although nitrous fumes are evolved, the mercury volatilises and combines with the iodine liberated by the nitrous fumes, thus preventing the formation of iodide of starch. The addition of bichloride thus raises the heat test without increasing the stability, rather the reverse, and therefore an insufficiently purified explosive may pass the heat test which would otherwise be rejected. Bichloride of mercury being prohibited, however, the value of the heat test is confirmed by experience, and is useful for nitro-glycerine and nitro-cellulose explosives. The usual test for mercury is to submit a small parcel of gold leaf to the vapour evolved at 70° to 80° C. and then test the leaf for mercury. Dr. Dupré does not find this satisfactory and recommends the spectroscopic test instead.—*Engineering and Mining Journal*, August 17, 1907, p. 298. (G. H. S.)

PRODUCTION OF ANHYDROUS BROMIDES. — " Reference has already been made in these notes (August 2, 1905) to the ingenious process by which Matignon and Bourrion succeeded in preparing the anhydrous chlorides of practically all the elements having refractory oxides, and the latter investigator has now modified the reaction so as to obtain the corresponding bromides. Most of the electro-positive elements are usually obtained in the form of their oxides, and Bourrion's method consists in heating one of these refractory oxides in a current of hydrogen bromide containing a small amount of sulphur chloride. In this way he has succeeded in converting the oxides of chromium, nickel, cobalt, and many of the rare earths into their anhydrous bromides. The products are thus obtained in a high state of purity, and are often distinctly crystalline. In view of the importance of the anhydrous metallic chlorides and bromides in many synthetical operations, this sulphur chloride process for producing these substances is a noteworthy improvement on the older method of treating the oxide with carbon and chlorine or bromine at fairly high temperatures."—*Times Engineering Supplement*, Sept. 18, 1905. (J. A. W.)

METALLURGY.

The Determination of Antimony in Hard Lead.—"Dr. H. Beckman describes in *Zeit f. angew. Chem.*, June 14, 1907, a novel and rapid method for the determination of antimony in hard lead. The method depends on the determination of the melting point of the alloy investigated, taken in connection with the cooling curve of lead antimony alloys.

The analysis is quickly carried out by melting some of the lead in a porcelain crucible, inserting a thermometer, and then waiting for the bath to solidify. The thermometer is made with two sets of graduations, the temperature being marked in black on the left of the capillary, and the corresponding per cent. of antimony being etched in red on the right side. One has only to read the per cent. of antimony corresponding to the position of the quicksilver when the alloy solidifies. The moment of solidification is easily determined, as the metal becomes distinctly viscid, a ring of solidified lead appears on the wall of the crucible, and the thermometer is moved around in the bath with some difficulty. An analysis can easily be carried out in five minutes and the amount of antimony can be determined accurately to the nearest 0·1 per cent.

As a general thing the thermometers are graduated only from 0 to 10 per cent. antimony. There are two percentages of antimony, as for instance 12 and 14 which have the same melting point. This fact, also the fact that with high percentages of antimony it is much more difficult to determine the melting point, makes the method uncertain when antimony in excess of 10 per cent. is present. The difficulty is, however, easily overcome by adding a definite amount of soft lead to the assay in those cases where it is known to be high in antimony, and after determining the percentage in the mixture, calculate the amount in the original sample."—H. Beckman. —*Engineering and Mining Journal*, Aug. 10, 1907, p. 262. (G. H. S.)

The Separation of Tin Oxide from Wolfram.—"The treatment of tin-wolfram ores by a magnetic separator and the extraction of wolfram from tin is now a fairly well-known process, but little has been said about the subsequent treatment of the wolfram.

Since the introduction of magnetic separation for Cornish ores, large quantities of tin oxide have been given away in the parcels of wolfram sold. Thus an analysis of a large parcel of wolfram, as prepared for sale at East Pool Mine, showed that it contained 28% of tin oxide and only 61% of wolfram, the remaining portion consisting of iron oxide, waste and moisture.

The authors have made a number of experiments upon that product of the magnetic separator known to dressers as tinny-wolfram, with the result that a large percentage of the tin can now be extracted. In order to carry out the experiments it was found necessary to devise a method of magnetically treating small quantities of the ore, quickly and without error or loss. To do this an electro-magnet was made.

The ore was passed under the pole of the magnet on a piece of sheet brass $\frac{3}{64}$ in. thick and 9 in. long by $2\frac{1}{2}$ in. wide, the 9in. sides being turned up $\frac{1}{2}$ in. A portion of the sample to be treated was thinly distributed over this brass tray, which was then carefully passed through the air-gap of the magnet in a direction at right angles to the edge of the pole piece. The magnetic particles were drawn up and held to the edge of the pole piece and the non-magnetic particles remained on the tray. On removing the tray a piece of paper was placed under the pole, and the current switched off from the coil, when the magnetic particles were released and dropped on the paper. In this manner quantities of 25 gm. could be readily separated in about 30 min.

A number of 25 gm. quantities of the separated tinny wolfram were taken in the dry state and separately boiled in the following solutions:—

1. Dilute sodium carbonate (10 gm. of powdered sodium carbonate to 100 c.c. of water).

2. Dilute iron sulphate (10 gm. of commercial iron sulphate crystals to 100 c.c. of water).

3. Dilute sulphuric acid (1 of strong acid to 20 of water).

The results of a second magnetic treatment of similar quantities, after boiling for three hours was as follows:—

No. 1 yielded 4·8 of tin oxide and waste.
„ 2 „ 6·7 „ „
„ 3 „ 32·5 „ „

These experiments showed that the sulphuric acid treatment was in the right direction. The next step was to compare the results obtained by using the common acids, sulphuric, hydrochloric and nitric.

Solutions of one to four were applied in a similar manner to the last set, with these results:—

Sulphuric .....34·5 of tin oxide and waste extracted.
Hydrochloric 29·5 „ „ „
Nitric ....Very little separation could be made.

These results show that sulphuric acid was the best for the purpose, and further experiments were made with sulphuric acid solutions of different strengths.

Strength of Solution.	Percentage Extracted.
1 in 60	24
1 „ 40	26
1 „ 20	32·5
1 „ 10	33·5

As these figures only showed the effects produced on small quantities (25 gm.) an experiment on a little larger scale was then made—1 cwt. of the ore was placed in a small kieve and covered with a 1 in 20 solution of sulphuric acid. In order to heat this, live steam was injected by means of a piece of india-rubber hose, and the whole was stirred up about every hour. This was continued for five days, the heating only being done during the daytime. Each morning a small quantity was taken out, well washed and dried, and passed under the electro-magnet, the following being the results:—

1st day......... 8·5% of tin could be separated.
2nd „ .... 10·5 „ „ „
3rd „ .... 17·5 „ „ „
4th „ .... 20·5 „ „ „
5th „ .... 20·75 „ „ „

As the percentage of tin oxide that could be extracted had stopped increasing, the acid was poured off and the ore thoroughly washed in a tye, dried, and passed through the Wetherill Magnetic Separator, which is used in the first separation of the wolfram from the dressed tin.

This gave:—

20·0% of tin oxide.
73·0 „ wolfram.
6·5 „ tinny-wolfram.
0·5% „ iron oxide.
100·0%

In view of these satisfactory results on a considerable quantity, further experiments are now being carried out with cheaper solvents, and with a view to perfecting the methods of heating and handling."—AMOS TRELOAR and GURTH JOHNSON.—*Institute of Mining and Metallurgy*, Sept. 19, 1907. (A. R.)

THE BLAISDELL PRESSURE FILTER. — A steel cylinder contains a series of filter leaves, the number of which is varied according to the capacity desired; connections are provided for vacuum and pressure pumps and for slime, water and solution.

The filter leaf consists of a series of non-porous columns provided with drainage grooves, on each side of which is a covering of canvas large enough to overlap the entire frame, and stitched so that no leakage can take place to the interior of the leaf. Charging tanks holding slime, wash water and wash cyanide solution enter into the filter by gravity and receiving tanks below the cylinder take the discharge. In operation the slime from the agitator vats is forced into the pressure cylinder, where the clear solution passes into the filter leaves and out through the discharge pipes to gold-solution tanks for precipitation, the slime remaining as a cake on the outside of the filter leaves. When the cake is about 2 in. thick the flow of pulp is cut off and the surplus remaining in the cylinder is returned to the storage tank. Wash solution is then introduced and forced through the cake, surplus solution being returned to its tank when washing is complete.

While these transfers of liquids are taking place, a partial vacuum is kept within the leaves, so as to make the cake adhere, and when washing is over, air or water under pressure supersedes the vacuum and causes the cake to drop off and be discharged through the bottom of the cylinder.

One of the filter leaves is supported by a weighing device, which shows the progress of formation of the cake, and an indicator connects with an electric bell when the proper thickness is reached. Seven different sizes of filter are constructed, with rated capacities varying from 12 to 500 tons of dry slime per 24 hours."—*Engineering and Mining Journal*, Sept. 7, 1907, p. 446. (G. H. S.)

ELECTRO-CHEMICAL AND METAL INDUSTRIES IN THE SOUTH OF FRANCE.—"The electro-chemical and metal industries have continued to make progress in 1906, which has seen an addition of some 5,000 h.p. for the production of aluminium and the same increase for carbide of calcium. These increases have been obtained by the completion of existing hydraulic works, but the present year (1907) will see the harnessing of further falls in the service of these industries not only in the Alps, but also in the Pyrenees. The use of what is fantastically called 'white coal' has attained a remarkable development in South-Eastern France within the last few years.

Aluminium.—The water-power used for the various chemical industries in 1906 amounted in the aggregate to 100,000 h.p., which may be distributed as follows:—

	Horse-power.
Aluminium	35,000
Carbide of calcium	25,000
Chlorate of potassium or sodium ...	15,000
Metallurgic products, steel	22,000
Soda, chlorine ... ... Hypochlorites	1,000
Various	2,000
Total	100,000

The success of the 'Société Grenobloise de Force et Lumiere' in bringing water-power 112 miles from Moutiers to Lyons has given birth to a scheme for bringing many thousands of horse-power from Bellegarde (Ain), on the Rhône, to Paris.

The production of aluminium has made rapid strides of late, having risen from 1,647 metric tons in 1904 to 1,905 metric tons in 1905, and though the actual figures of production for 1906 are not yet accessible, the export alone increased by over 50 per cent. as compared with the previous year; so that it is certain that the rate of progression in production was at least maintained. The extraction of bauxite, the raw material of aluminium, has also risen from 75,000 metric tons in 1905 to 103,000 metric tons in 1906. A large part, however, of the quarried bauxite is of no use for the manufacture of aluminium, and is employed for making fireproof ware.

Nitrates.—The production of nitrates is about to be revolutionised by the manufacture of cyanamide and nitric acid in an electric furnace. Cyanamide is obtained by the action of nitrogen upon carbide of calcium at the temperature of the electric furnace. This substance turns in the soil into salts of ammonia, and can thus enter into competition with the sulphates of ammonia produced in gas and sewage works. A factory for cyanamide is being established in the district of the Tarentaise.

The production of nitric acid in the electric furnace by the direct union of oxygen, nitrogen, and steam is carried on in a Norwegian factory. The nitric acid, neutralised by lime, gives solid nitrate of lime, which may be advantageously used instead of nitrate of soda for agricultural purposes. When the fact is borne in mind that France alone imported 241,272 metric tons of nitrate of soda in 1906, it is clear that if only the nett cost can be kept sufficiently low, this new industry has a great future before it. So far, only samples of cyanamide and nitrate of lime have reached France, but it is announced that important works are being organised in Germany and Italy for the production of cyanamide, and in Norway for nitrate of lime."—Mr. CONSUL VICARS, "Report for 1906."—*London Mining Journal*, Sept. 21, 1907, p. 374. (A. R.)

THE USES OF BAUXITE.—"Bauxite has proved to be of great value as a basic refractory material in that it resists well the scouring action of metallic oxides in the furnace. Bauxite is used most conveniently in the form of bricks for this purpose. The Berger patent covers a process for making these bricks. Natural high-alumina low-silica bauxite from Arkansas is used, bound by a small percentage of plastic fireclay, sodium silicate, or lime. The bricks contain from 88 to 90 per cent. of alumina and from 10 to 12 per cent. of ferric oxide, silica, and titanic acid. The percentage of silica can be reduced to 6 or 8 per cent. by the use of a bond free from silica, so that the brick is not appreciably detrimental to basicity. Bauxite brick seems to be especially adapted for the linings of basic open-hearth steel furnaces. The highest grade is used in the floor and walls up to the slag line, protected by a bed of calcined bauxite. Above the slag line cheaper brick with a lower percentage of bauxite can be used.

A severe test was made some time ago at one of the basic open-hearth furnaces of the Bethlehem Steel Works, in which a bauxite brick and a magnesite brick were placed side by side near the gas and air ports, and thus were subjected to the highest temperature attainable in the furnace. The magnesite brick bent and showed viscosity after seven minutes, against fifteen minutes for the bauxite brick. A bauxite brick and a magnesite brick were then submerged in slag near the doors for some time, after which they were withdrawn and examined when cold. The magnesite brick was incorporated with slag, while the bauxite brick, when broken, showed that the slag had not penetrated to its centre, but had remained as a coating over the outside.

In addition to its use for open hearths, bauxite brick has proved successful for other uses, two of which are as a lining for rotary Portland cement kilns and as a lining for lead-refining furnaces. Used as a lining for the hot zone (10 to 12 ft.) of a 60-ft. rotary cement kiln, 6 in. bauxite bricks have been found to give more than ten months' continuous night and day service. This indicates that the material is superior to fire brick, and that the saving of output that would otherwise be lost during frequent shut-downs for the purpose of relining or patching a kiln in the hot zone will more than compensate for the increased cost of superior lining.

In lead-refining it has been found that the scum composed of highly basic oxides readily attacks the free silica and silicates of alumina in a fire brick and rapidly decomposes the latter, whereas the basic bauxite lining, theoretically and in practice, is better adapted for use in such furnaces. Tests show that bauxite brick really lasts five or six times as long as the fire brick lining.

Bauxite bricks are manufactured by the Laclede Fire Brick Company, of St. Louis, for the American Bauxite Company, and their cost is considerably less than that of magnesite brick.

It is noteworthy that the attention of the public was first called to the Arkansas bauxite deposits in 1891 by Dr. J. C. Branner, then State Geologist of Arkansas. In 1895 their commercial value began to be appreciated. In 1899 were reported the first statistics of production from this field, and reference to the table will show that the production of 1906 has exceeded the most sanguine predictions."—ERNEST F. BURCHARD, "Mineral Resources of the United States."—*London Mining Journal*, Sept. 21, 1907, p. 273. (A. R.)

THE NEW ALLOYS.—"During the last three or four years a great number of new metallic alloys, with very varied properties, have appeared upon the market, amongst which we shall select the most important, and give their composition.

Cupro-Magnesium (Cu = 90, Mg = 10) is utilised as a deoxidising agent for unrefined copper, in the proportion of 1%. The copper thus obtained has an electric conductivity less than that given by silicium for deoxidising, but deoxidisation is very easy by mere fusion, and very economical.

Phono-Electric Wires (Cu = 98·55, Sn = 1·40, Si = 0·05) are employed for telephones and trolleys. They withstand the effects of traction much better than pure copper, though their electric conductivity is only two-fifths that of pure copper. In preparing this alloy silicium is totally sacrificed. Consequently it does not exist beyond traces in the alloy.

Sterline (Cu = 68·52, Zn = 12·84, Ni = 17·88, Fe = 0·76) is a kind of white metal; a substitute for silver.

Manganese Resistance Metal (Cu = 85, Fe = 3, Mn = 12) is a substitute for German silver, especially for resistance boxes for electric measurements; specific electric resistance is only 3-100ths to 4·5-100ths that of copper.

Manganine (Cu = 82·12, Ni = 2·29, Fe = 0·57, Mn = 15·02) is also utilised for resistance boxes, and owes to the presence of nickel a very high fusion point and extremely low coefficient of temperature.

Acid Proof Metal (Cu = 82, Zn = 2, Sn = 8, Pb = 8) is particularly useful for paper works where the bisulphite process is employed for manufacture of pulp. As a matter of fact, it is only proof against weak or diluted strong acids, except nitric, which quickly attacks it.

Victor Metal (Cu = 49·94, Zn = 34·27, Ni = 15·40, Al = 0·11, Fe = 0·28) is whiter than German silver, which it can often replace, though less easy to work. It perfectly withstands the effect of salt water and air. Consequently it is chiefly utilised for marine engines.

Aluminium Silver (Cu = 57, Zn = 20, Ni = 20, Al = 3) is a white, very tenacious metal, which remains bright in air, and advantageously replaces steel whenever there is danger of rust.

Tempered Lead (Pb = 98·51, Sb = 0·11, Sn = 0·08, Na = 1·3) is manufactured by placing small fragments of sodium in the molten metal. This alloy is not so soft as lead, and can be rolled into thin sheets without tearing. When percentage of sodium is rather great, tarnishing is prevented by coating the metal with paraffin. Thus formation of soda is prevented, owing to oxidation of the excess of sodium by atmospheric oxygen. For this reason it is valued for manufacture of shaft bearings, because the soda formed, as the bearings wear away, saponifies the lubricant and produces a soap which acts even better than the oil.

Alkali Proof Metal is iron with 5% to 10% nickel. All alloys containing zinc, tin, lead, aluminium, antimony, or silicium are readily attacked by caustic alkalies."—*La Nature.*—*London Mining Journal*, Sept. 21, 1907, p. 361. (A. R.)

INFLUENCE OF IRON IN COPPER ELECTROLYSIS.—"The influences of iron in copper electrolysis may be summed up as follows:—

1. High percentages of iron in the ferric condition materially retard copper deposition. Ferrous salts have little or no effect.

2. The effect of the electric current in the usual solution is to reduce ferric salts to ferrous. This reduction must proceed to a certain point before all the copper will remain on the cathode. This point is about the same whatever be the iron and copper percentages, but it varies some as the solution is circulated or stationary.

3. Rapid circulation of the solution by a stirrer permits the use of high current densities without sponging, thereby making deposition faster. The current also is more efficient for both depositing copper and reducing iron."—E. L. LARISON.—*Engineering and Mining Journal*, Sept. 7, 1907, p. 442. (G. H. S.)

VALUES IN GOLD SLUDGE DUST.—"A few particulars concerning the working of a device for saving values lost by dusting during the preparation of zinc-box precipitates for smelting are given by the *Journal of Mines of Western Australia*, June 29, 1907. A hood, suction fan and mechanical dust-collector system was installed in the clean-up room at the Great Boulder Perseverance mine at a cost of $300 and subsequently operated as required during the period from Oct. 1, 1906, to June 8, 1907. All screening and mixing of sludges was done under a pyramidal-shaped hood, fitted with side curtains reaching to within a few inches of the floor level. By means of the fan the dust was drawn upward, sucked away through an opening in the top of the hood, and delivered to the dust collector, from which the recovered material was removed at the end of the period indicated. The quantity of dust recovered was 16·3 lb., containing 29·756 oz. of fine gold, valued at $615. The total weight of sludge handled during the eight months' period was 20.784 lb., and from this 46.790 oz. of fine gold were recovered by smelting. The recovery of gold from the dust, though insignificant when compared with the total values handled, justified the cost of installation of the system."—*Engineering and Mining Journal*, Sept. 7, 1907, p. 443. (G. H. S.)

MINING.

Eight Hours' Day.—"The final report of the Departmental Committee, appointed to enquire into the probable economic effect of a limit of eight hours to the working day of coal miners, will be read with interest both by those who view the question from the economical standpoint, and those whose concern with it is mainly of a social character. The present duration of the working day varies greatly with different districts, but the examination of statistics shows that, making allowance for customary weekly or fortnightly total stop days and short days, the average theoretical full week's work amounts to seven minutes short of 50 hours. Assuming the continuance of the short and idle days undiminished, a legal eight hours' day would effect a further reduction of 10·27 per cent., and it is obviously an easy task to calculate the diminution in output which, with constant labour efficiency, this would entail. There are, however, compensations which would to a large extent neutralise any such reduction. Owing to stoppages at the collieries for various reasons, the actual falls short of the theoretical working week by more than six hours, and it is believed that under a legally restricted eight-hour day a portion of this time would be utilised. Further factors of adjustment would be an improvement in the efficiency of the labour, more especially in districts now working beyond the average time, and an extension of the use of labour-saving machinery. Although expressed in guarded language, the opinion of the Committee seems to be that while an eight hours' day would occasion some reduction in production, its extent would fall substantially short of the theoretical maximum. As to the permanent results of a legally restricted day for workers in coal mines upon wages and employment, these are stated to be 'impossible to see,' and 'imprudent to forecast.'"—*London Mining Journal*, Aug. 31, 1907, p. 277. (J. Y.)

Rescue Apparatus for Use in Coal Mines.—"Refuge chambers have been suggested from time to time, to be built in different parts of the mine, the chambers being closed against the noxious gases present in the mines, after the men have entered them, and being supplied with air and milk, etc., for sustenance from the surface, but not much appears yet to have been done in that direction. The difficulties in the way of the establishment of refuge chambers are very great. A large number of them would be necessary for large mines, such as rule at the present day, and the constant 'working' of the mine would necessitate a somewhat heavy outlay to keep them in order.

Apparatus, however, that can be used by the rescue parties are far more practical. The great difficulties which rescue parties have to encounter, are the poisonous atmosphere resulting from conversion of a large portion of the oxygen of the atmosphere of the mine, into carbonic acid, and carbonic oxide, and the fact that the explosion very often seriously damages the roadways. With the rescue apparatus described, however, a great deal may be, and has been done, to enable explorers to penetrate into even very poisonous atmospheres. At one of the meetings of the Institute of Mining Engineers last year, two apparatus were described, one by Herr G. A. Meyer, of Westphalia, whose apparatus and whose corps of trained men had been employed in the rescue work, after the unfortunate disaster at the Courrières, the other by Mr. W. E. Garforth, the Chairman of Messrs. Pope & Pearsons, a large firm of colliery owners in Yorkshire.

The principle upon which both apparatus, and, in fact, all apparatus designed for the purpose, are arranged, is the provision of a supply of fresh oxygen to the lungs of the explorer, and the carrying off and re-oxygenation of the air that he exhales at each respiration. Both apparatus include a supply of oxygen and a supply of a substance that will absorb the carbonic acid, and allow the air that has been expired to be used over again. There are several difficulties in the working out of an appliance of this kind. One is the weight of the apparatus which has to be carried, containing the oxygen and the absorbent of carbonic acid, and another is the prevention of the polluted air of the roadway in which the explorer is moving, reaching the lungs. In Herr Meyer's apparatus the supply of oxygen compressed to 1,764 lb. per sq. in., is carried in two flasks on the back of the explorer, together with a pressure gauge. On his breast the explorer carries an india-rubber bag, with a second smaller india-rubber bag inside it, the latter containing the absorbent. In the lower part of the large india-rubber bag is also carried an absorbent for moisture. The absorbent of carbonic acid in Herr Meyer's apparatus is caustic potash, 1 kgm., or 2·2 lb., being carried, this being stated to be sufficient for two hours' work. A safety or relief valve is also fixed on the apparatus, to guard against the possibility of the pressure of oxygen in front of the mouth, being too great for the breath to expire properly, and therefore to an accumulation of carbonic acid being formed in the lungs, leading to asphyxiation. A tube enters the mouth, and the nostrils are plugged with cotton wool, soaked in a special kind of grease. The oxygen is supplied to the lungs through a reducing valve, and the expired breath is made to draw in a mixture of oxygen and regenerated air from the absorbing apparatus, by the application of the injector principle. An electric bell, ringing every five minutes, warns the explorer when half the oxygen has been consumed, so that he may at once commence his retreat. In Mr. Garforth's apparatus, two cylinders of oxygen are also carried, one under each arm, the oxygen being compressed to 1,800 lb. per sq. in., and the bottles being rounded to the shape of the body, so as to be carried conveniently. The absorbent, which is also caustic potash, is carried in a bag on the back. The mouth and eyes are protected by a conical shaped vessel fitting over them, arranged so, that the moisture from the breath does not accumulate on the eye pieces, and breathing can take place from both mouth and nostrils. A helmet completes the apparatus in both Herr Meyer's and Mr. Garforth's arrangement, and the expired air, in both forms of apparatus, passes from the mouth through the absorber, where it is deprived of its carbonic acid, and then returns to the mouth, taking up a small portion of oxygen from the reservoir, on its way.

In Mr. Garforth's apparatus the supply of oxygen is controlled automatically by a reducing valve, worked by the lungs themselves. The quantity of oxygen required was found, by actual experiments, to vary very considerably when at rest and when at work. From experiments carried out in the gallery at Altoft's colliery (Messrs. Pope and Pearsons), with Herr Meyer's and other apparatus, men were able to remain in the colliery for 1 hour and 40 minutes when not exerting themselves, but only from 15 to 20 minutes during exertion, such as would be required in the act of exploring, climbing over falls, through air passages, and so on. Mr. Garforth also found, from the experiments, that if the supply of oxygen was the same, whether a man was exerting himself or not, very unpleasant feelings were produced, in some cases vomiting taking place, and hence he came to the conclusion that it was absolutely necessary that the quantity of oxygen taken in should be in proportion to the work done. Further, he came to the conclusion that the explorer should have both hands free, and that neither hand should be engaged in regulating a valve controlling the supply of oxygen. In his latest apparatus he provides a linen jacket, weighing 2 lb., buttoning close up to the throat, the pipes from the oxygen cylinders passing through slits in the jacket, up to the mouth-piece apparatus, and he has an electric lamp with a small accumulator and a flexible cord, arranged so that the explorer can either hook the lamp through one of the button holes of the jacket, or carry it in his hand. The whole of the apparatus, including the linen jacket, weighs about 30 lb., distributed over the head, shoulders and hips. It can all be put on and taken off without assistance, the oxygen cylinders being carried in the side-pockets of the jackets, and warning being given to the explorer when one cylinder is exhausted, so that he can at once commence his retreat. Some interesting analyses are given by Mr. Garforth, of the gases in his rescue apparatus, the analyses having been confirmed by Dr. Haldane, who has done so much in the matter of the sanitation of underground workings. The following is one analysis :—

	Oxygen. Per cent.	Carbonic Acid. Per cent.	Nitrogen. Per cent.
In the mouth-piece ...	17·9.	3·38.	78·72.
Just before passing through the absorber ...	16·0.	3·4.	80·6.
In the bag or receiver before entering the mouth-piece	19·0.	0·03.	80·97.

It will be seen that the carbonic acid in the expired air from the lungs, is entirely absorbed by the potash, but that the percentage of oxygen is slightly less after passing through the absorber than that in atmospheric air, the difference being made up from the oxygen supply in the cylinders.

Several tests were made in the gallery at Altoft's colliery, in which coal miners and colliery officials remained in a noxious atmosphere for several hours together, in the latest experiment a miner having remained there for six hours.

There appears to be a difference of opinion as to the matter of the supply of oxygen. On the Continent it appears to be held that there is no harm in having an excessive supply, that the lungs will only take in what they require. Mr. Garforth's experiments, it will be seen, negative this conclusion. There appears, however, to be no difference of opinion as to the effect of an excess of carbonic acid and an absence of oxygen. Breathing becomes unpleasant when 4 per cent. of carbonic acid is present in the atmosphere, it is difficult at 6 per cent., and at 10 per cent. asphyxiation commences. No amount of excess of oxygen will neutralise the excess of carbonic acid. With insufficient oxygen breathing is unpleasant with only 12 per cent., and is very difficult at 6 per cent.

One important point should be noticed, that is mentioned by Mr. Garforth. The oxygen in the cylinders should be absolutely pure. It has been stated by Dr Haldane that cylinders of oxygen sometimes contain as much as 10 per cent. of nitrogen, and this may lead to the asphyxiation of the explorers.'—*Indian and Eastern Engineer*, August, 1907, p. 53. (A. R.)

THE DESIGN AND EQUIPMENT OF SHAFTS FOR DEEP WINDING.—"During the last decade the subject of deep mining has been brought prominently before the mining engineers of the country owing to the rapid exhaustion of the seams which have hitherto been worked at moderately shallow depths, and shafts are now being sunk and seams opened out at depths which are very much greater than had to be dealt with a few years ago.

At one or two of the older collieries the lower seams have been opened out as the upper seams have been exhausted, and the best has been made of the existing plant and machinery ; and although coal has been, and is, worked at great depths under those conditions—and, no doubt, with satisfactory results—it nevertheless follows that there is great scope for methods being adopted for the winning and working of seams at 800 yards in depth and over. I take 800 yards as an illustrative figure, as I am of opinion that any shaft for the purpose of working coal below that depth may be considered a deep one.

There are so many and varied factors to be taken into consideration that it is almost impossible to deal with them except in a very general way.

First, size of shaft. If a large acreage of coal has to be worked, it is of advantage to have the shafts as large as possible ; first, in order to give the utmost possible scope for a maximum quantity of air being circulated through the mines, as the greater the depth the more necessary it is for such air being divided into splits, with the object of cooling down the various sections of the workings to the utmost extent ; and secondly, large shafts enable cages of larger size to be introduced, and the number of tubs raised at one time to be appreciably increased, although with the appliances that are now available for simultaneous decking the question of the size of the cages is not all-important.

In deciding upon the size of the shafts, the character of the strata to be sunk through is an important factor ; if it should be such as to preclude the sinking of the pits by normal methods and compel the introduction of sinking by tubing being forced down by hydraulic pressure in order to deal with running sand and water, or other difficulties, then the size of the shaft is limited. I am personally of opinion that large shafts are of advantage in dealing with a large quantity of coal, and I have recently put down at a depth of 760 yards shafts 23 ft. in diameter, and a number during the last 20 years all over 18 ft. diameter.

It is convenient here to mention the question of guides. These can be either rigid steel guides or rope guides. I have rope guides working perfectly satisfactorily in pits over 900 yards deep.

The question of the type of engine to be adopted, and especially the drum, affords a wide field for ingenuity. High pressure steam-engines seem to me to be best adapted for coping with the problem before us, and with the introduction of low pressure turbines, which afford a means of utilising the exhaust steam, they present such advantages in the way of simplicity that I think they will prove themselves superior to compound engines.

The next question is the drum. A parallel drum with a balance-rope works economically, but the depth at which it can be used is limited, owing to the impossibility of getting a satisfactory lead for the rope off the drum on to the pulley when large ropes and great depths have to be dealt with. Several methods have been adopted for overcoming this difficulty, namely :—

The Koepe system.
The conical drum.
The flat rope drum.
The modified conical drum and balance-rope.

Winding-ropes to raise a load of 15 to 20 tons have to be very large. At the present time for a depth of 760 yards I have a locked coil rope 5½ in. in circumference raising a load of 15 tons, the weight of the rope itself being 9 tons.

If the shaft is deepened, the question of the size of the rope, if the same load has to be raised, is a matter of considerable difficulty, especially if a ten to one margin between the working-load and the breaking-strain is adhered to.

I have considered the desirability, where the size of the shafts would allow it, of having four cages in a shaft and two sets of winding-engines, each raising half the load that I have spoken of, so as to keep the size and weight of the rope within reasonable limits.

The late Mr. David Davy, of Sheffield, was at work on a scheme at the time of his death for winding by means of an endless chain, hanging on the tubs one at a time by means of cages which formed part of the chain itself. It was a very ingenious arrangement, but as to its practicability I should not like to express an opinion.

The next point is the question of ventilation, and, as I have said at the commencement of this Note, a maximum volume of air is all-important, so as to afford facilities for dividing the air into a number of splits, each split dealing with its own separate area of working-faces, the collieries being laid out as far as practicable in independent sections. This is the only method by which the working-places at, say, depths of 1,200 yards can be kept so as to enable men to work with reasonable comfort. The question of ventilation may not be actually within the four corners of the subject I am asked to bring before you, but in deciding upon the size of the shafts it has to be taken into account, and therefore I mention it." — CHARLES EDWARD RHODES. — *Indian and Eastern Engineer*, Sept., 1907, p. 106. (A. R.)

SINKING OPERATIONS THROUGH QUICKSAND.—"The work done in sinking Bentley Colliery was described in some detail, particular attention being paid to the method adopted in sinking through the 50 ft. of quicksand, which was met with near the surface in the case of both pits. This was to lower cast iron tubbing bolted together with steel piles, grooved and tongued into each other, forming a complete circle round the outside of the bottom ring and sliding on the back. The piles were pushed downward by pressure applied against the bottom flange of the bottom ring. The tubbing was then lowered down, sliding inside the piles, and additional rings of tubbing were added at the top. The tubbing was supported at the top by screws with cast steel shoes bolted to the top ring, the shoes being attached to vertical girders built into the sides of a cement concrete block, 6 ft. thick, 50 ft. square, and with a hole 23 ft. in diameter in the centre. The tubbing, 20 ft. in inside diameter, 6 in. wide on the bed, and 1 in. thick was fitted with inside flanges and bolted together with bolts ⅞ in. in diameter. The piles were built of mild steel plates 1¼ in. thick and 8 ft. long, grooved and tongued into each other with a T iron, the whole being bolted together with counter-sunk bolts, and a bolt was inserted through a T slot in the centre to keep the piles close to the back of the tubbing. Ninety-six piles formed a complete circle, fitting closely against the tubbing. The segments of tubbing were made 1 ft. 9 in. deep, 8½ in. wide on the bed and 1½ in. thick, and bolted together with sheeting ⅜ in. thick between each joint. There were 12 segments in each ring, and each ring was stayed diagonally with four tie rods 1¼ in. in diameter. The strong cast iron guide ring fitted inside the piles below the tubbing, and ring of tubbing built upon the guide ring, were lifted by hydraulic jacks and bolted to the lowest ring of the tubbing. The guide ring weighed 17 tons, the piles one ton each, and a ring of tubbing 8¾ tons.

The President asked whether there were local conditions which prevented the adoption of the freezing process for sinking. He would like to have a comparison of the cost of sinking by the freezing process and by that which had been successfully adopted. Mr. J. W. Fryar, in his reply, said he had worked out the cost of the method adopted as compared with the ordinary method, and it had cost about £12,000 more to sink the two shafts through the quicksand than if the usual method had been adopted, but this included the cost of failures that occurred at the first shaft, and, therefore, was not exactly a fair comparison."—J. W. FRYAR and ROBT. CLIVE, *Times Engineering Supplement*, Sept. 11, 1907.—Conference, Institute of Mining Engineers, Sept. 4, 1907. (J.A.W.)

RAPIDITY OF THE DETONATION OF EXPLOSIVES.—"In a communication to the 'Académie des Sciences' M. Dautriche records the results of experiments in regard to the rapidity of the detonation of explosives. He finds that the velocity is a function of the density of the explosive, at first increasing up to a certain density △ ft., and then decreasing. The 'sensibility' of the explosive, defined by the weight of fulminate of the detonator necessary to detonate a cartridge, is sensibly constant in the ascending part of the velocity-density curve; it decreases rapidly, on the contrary, in the descending portion of the curve, so that for higher densities than △ ft. it is necessary to increase the detonating charge in order to avoid misfires. The *brisance* (shattering effect) of the explosive, measured by the lead block test, has no absolute relation to the velocity of detonation, so that the lead block test gives results almost independent of density, that is to say the velocity of detonation. M. Dautriche has also measured the velocity of the shocks communicated to the air by the detonation of explosive cartridges, and has found that these velocities reach up to nine times the velocity of sound." — M. DAUTRICHE. — *Indian Engineering*, Sept. 7, 1907, p. 155. (A. R.)

ROOF WEIGHTS IN MINES.—"The author pointed out that in underground mining the nature of the roof determined to a large extent the method of working to be adopted, and as seams were worked under widely different roof conditions, the support and control of the roof in mines opened a wide field for careful observation. Of the many conditions requisite for success in longwall working, that of obtaining a uniform settling of the roof behind the working face was one of the most important, for when the desired result was attained the bending roof became a lever which assisted materially in the work of coal getting. Roof weights might be divided into three classes, the first weight, limited to the working face; the second weight, the goaf weight following the working face; and the third weight, the general subsidence of the strata. Although the first weight was mainly due to the weight of the subsiding roof, there were other influences at work, one of which was the lateral compression of the strata, which accounted for side weight in the roads, and also for the fact that the working in an upper seam disturbed the roads previously made in a lower seam. Dealing with the second weight, he pointed out that as the coal face advanced the roof bent in the goaf due to the first weight, but the effect of the second weight was mainly seen in the crushed packs and reduced height of the roads, necessitating their re-ripping or re-dinting. During the period of second weight the roof should be allowed to settle on the packs as uniformly as possible. Wood packs put in the gate sides and middle props in the roads should be avoided, as they tended to break and cut the roof. When the coal face was abandoned the second weight advanced towards the face until it practically reached it. The conditions during the period of second weight were the reverse of those for first weight. The advancing line of settled goaf formed the fulcrum of the roof lever, and the power was applied by the weight of the higher strata subsiding towards the retreating coal face." — H. T. FOSTER, *Times Engineering Supplement*, Sept. 11, 1907. Conference of Institute of Mining Engineers, Sept. 4, 1907. (J. A. W.)

DEEP LEVEL TEMPERATURE AND VENTILATION.—"Readings taken in the New Chum and Victoria mine, Bendigo, by Mr. A. H. Merrin and Mr. W. Abraham, show that at 3,357 ft., or the bottom plat, the thermometer registered 78½°, while the aerometer showed that 1,360 cub. ft. of air was passing down the shaft per minute. The air is returned through a fourth compartment, which also allows of blasting fumes being readily taken away. In a centre-country winze below 3.357 ft., at 160 ft. from the shaft, the temperature was 79½°, while the air registration was 1,400 cub. ft. per minute. The temperature at a depth of 2,200 ft. was 75°, and the volume of air 5,100 ft. per minute."—*Colliery Guardian*, Sept. 3, 1907, p. 495. (A. R.)

MISCELLANEOUS.

SALINE SOLUTIONS ON DUSTY ROADS.—"In a recent number of the *Bulletin de la Société Industrielle de Rouen*, Messrs. Houzau and Leroy call attention to the use of solutions of the chlorides of sodium, calcium, and magnesium with the object of laying the dust produced by automobiles. The two latter chlorides are especially advantageous owing to the deliquescent nature of these salts. Calcium chloride was first tried for this purpose as long ago as 1828. A solution of this salt having a specific gravity of 10-15°. Beaumé is a suitable strength for sprinkling, and is applied to the roads in the same way as water. This salt has the further advantage of being cheap and non-poisonous, having in fact certain disinfecting properties; it is not corrosive to ordinary metals, and may therefore be stored in metallic vessels, but it attacks the unpainted surfaces of copper, nickel, and brass. Sprinkling with tar or oil is more expensive than with calcium chloride, and this salt does not give rise to dark mud, rendering the roads slippery."—*Times Eng. Supplement*, Sept. 25, 1907. (J. A. W.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

THE MINERAL INDUSTRY. Vol. XV. 1906. 21s. (New York City, U.S.A.: The Hill Publishing Company.)

"*The Mineral Industry* is too well known to make any detailed criticism of the issue lately to hand necessary. The current volume, which so far as composition is concerned dates from the end of May last, follows the lines of recent numbers, and is practically a precis of the literature bearing on the period covered. At a time when it becomes more and more a study in itself for the ordinary professional man to keep himself posted in the current technical literature of each year, the release from so much mechanical labour is of the utmost value; and it is in this light that we think the value of our contemporary's compilation should be considered. Other arrangements of the task of dealing with the history of each year's mining and metallurgical progress may at first sight seem more attractive, but we question if in the end any would be found of as great a value by the ordinary practitioner as this, which leaves him free to devote his own time to any subject of specialisation or matter demanding reflection rather than familiarity with multiplicity of detail.

In conclusion, we need only say that anyone intelligently considering the enormous scope of the work, and recognising on what subjects an American publication is naturally best informed—in a word, using the matter with discretion—will find the greatest assistance from the current volume, especially where, as is the case with most mining men, works of reference of a more specialised kind are beyond his reach."—*London Mining Journal*, Sept. 21, 1907, p. 361. (A. R.)

THE COPPER MINES OF THE WORLD. By WALTER HARVEY WEED. Large 8vo. xiv. and 375 pp. (New York and London: Hill Publishing Company, 1907.)

Mr. Weed was for over twenty years on the United States Geological Survey, which country produces 55% of the world's output of copper, and as he has seen practically all the big mines of Canada and Mexico as well, the book could scarcely fail to become the standard work on copper deposits.

The high expectations concerning it are fully justified on perusal of the contents, which are classed under two heads: 1. Geology of Copper; 2. Description of the Principal Copper Mines of the World; under both of which most valuable information is given, largely the result of the author's own observation.

Of particular interest are the sections on "Outcrops and Gossan Formations," and "Genesis of Copper Deposits," in which the results of recent work in this department of geology are clearly set forth.

The printing is excellent, and the book contains no less than 159 illustrations, ranging from microphotographs of copper bearing rocks to maps of copper districts. It is, therefore, distinctly a book to be read by all who wish to be up to date on copper occurrences.—(G. H. S.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 420/07. C. Neumann. Improvements in air compressors. 27.9.07.

(C.) 421/07. J. M. Adams. An apparatus for carrying paraffin tins and the like and to enable such tins to be used as buckets. 28.9.07.

(P.) 422/07. W. H. Angus (1), S. Martin (2). Improvements relating to amalgamated plates employed in the recovery of metals from ores. 28.9.07.

(P.) 423/07. O. Rasmussen. Improvements in tamping plugs. 3.10.07.

(C.) 424/07. C. P. Stewart. Process for making sugar. 4.10.07.

(C.) 426/07. J. Russell (1), J. H. Mulligan (2). Improvements in apparatus for recording bell or like signals specially applicable to mines. 4.10.07.

(C.) 427/07. G. Schultz. Improvements in the production of aromatic nitro-compounds from solvent naptha, and in the use of the same for producing safety explosives. 4.10.07.

(P.) 428/07. W. C. Stephens. Coupling device for connecting hose fittings to rock drilling machines and for analogous purposes. 4.10.07.

(P.) 429/07. E. J. J. Saville. Improvements in hose couplings. 4.10.07.

(P.) 430/07. H. Zerning. Improvements relating to the manufacture of filaments for electric incandescent lamps. 4.10.07.

(P.) 431/07. J. H. Hughes (1), R. H. Harriss (2). An improved means of disintegrating and rendering friable diamond bearing material by the application of chemicals and gases. 5.10.07.

(P.) 432/07. J. van Waart. A device or construction of a water wheel with double boxes for discharging water and other fluids with or without producing power. 7.10.07.

(P.) 434/07. J. W. Cowie. Improvements in means for driving ore feeders of stamp batteries. 9.10.07.

(P.) 435/07. N. J. Sundin. Automatic safety-catch and brake for lifts and hauling skips. 9.10.07.

(C.) 437/07. R. Stanley. Means and method of crushing and treating ores for the extraction of precious metals. 11.10.07.

(P.) 439/07. J. Crause (1). M. Cohen (2). Improvements in the chemical treatment of slimes and ores. 16.10.07.

(P.) 441/07. E. O. Blackmore. Improvements in or relating to shaking chutes for conveying ore in mines. 18.10.07.

(C.) 442/07. Dr. Novak (1), Dr. R. Escales (2). Improved process for accelerating the separation of nitro-glycerine from acids. 19.10.07.

(C.) 443/07. O. Hetlesaeter. Improvements in excavators. 19.10.07.

(C.) 444/07. E. Bellini (1), A. Tosi (2). System of directed wireless telegraphy. 19.10.07.

(C.) 445/07. W. E. Morton. Improvements in the means for lubricating the pins of cranks or similar moving parts of locomotive or stationary engines and the like. 19.10.07.

(P.) 446/07. A. B. E. W. Richards (1), Executrix dative of J. W. Richards (2). Improvements in machines for the manufacture of cupels and the like. 21.10.07.

(P.) 448/07. J. Nicholson. Improved centrifugal apparatus for filtration or separation of liquids from solids. 23.10.07.

(C.) 449/07. The Butlin Gear, Ltd. (1), G. L. Butlin (2). Improvements in power transmission gearing. 24.10.07.

(P.) 450/07. W. C. Stephens. Improvements in rock drills. 24.10.07.

(C.) 451/07. W. O. Pelt. Process for compressing, purifying, drying and cooling air and other gases. 25.10.07.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

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BLACK, JAMES, *to* Buck's Reef Mine, Colleen Bawn Siding, Rhodesia.
BRICKHILL, H. G., *l/o* Brakpan; Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston.
BROOKS, E., *l/o* Jupiter; P. O. Box 108, Krugersdorp.
DAVIS, R., *l/o* Queque; Waterfall Syndicate Mill, Penhalonga, Rhodesia.
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HYLAND, L., *l/o* Hartley; Gatooma Station, Rhodesia.
JACKSON, G. T., *l/o* Egypt; West View, Finningley, Doncaster, England.
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PATTERSON, P. F., *l/o* Lomagundi; c/o Messrs. Turrell & Patterson, Mining Engineers, Penhalonga, Rhodesia.
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WILSON, J. K., *l/o* Lydenburg; Angelo G. M. Co., Ltd., East Rand.
WISDOM, G. E., *l/o* Johannesburg; Riverlea Mine, Queque, Rhodesia.

THE JOURNAL
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Chemical, Metallurgical and Mining Society
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VOL. VIII. DECEMBER, 1907. No. 6.

Proceedings
AT
Ordinary General Meeting, December 21, 1907.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, December 21st, Prof. J. Yates (President), in the chair. There were also present :—

30 Members: Dr. J. Moir, Messrs. T. L. Carter, R. G. Bevington, K. L. Graham, A. McA. Johnston, A. Richardson, A. Whitby, H A. White, Prof. J. A. Wilkinson, W. Beaver, A. A. Coaton, A. D. Gilmore, J. Gray, H. D. Griffiths, F. G. Guthrie, C. B. Kingston, E. Lawrie, G. A. Lawson, W. D'A. Lloyd, W. P. O. Macqueen, J. P. McKeown, J. T. Milligan, T. T. Nichol, O. D. Ross, C. B. Saner, M. Torrente, A. D. Viney, J. Watson and J. O. Welch.

9 Associates: Messrs. S. J. Cameron, J. C. Chilton, J. Cronin, W. J. N. Dunnachie, J. H. Harris, G. F. Jones, R. W. Leng, A. M. Thomas and W. Waters.

9 Visitors and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as published in the *Journal*, were confirmed.

NEW MEMBERS.

Messrs. Beaver and Nichol were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been duly elected, as follows :—

FITCH, FRANK, Black Mountain Mining Co., Magdalina, Sonora, Mexico. Cyanide Superintendent.

HOLLOWAY, GEORGE THOMAS, 57, Chancery Lane, London, W.C. Metallurgist.

JANSEN, PETRUS JOHANNES, Moeara Aman, Benkoelen, Sumatra. Mining Engineer.

NIS, EDWARD A., New York and Honduras Rosario Mining Co., San Juancito, Honduras, Central America. General Manager.

RICHARDS, FRED., M.I.M.E., "Blaenau," Compton Road, Canonbury, London, N. Director of Mining Companies.

The President: I think it is worth while drawing attention to one point in connection with the new members you have elected this evening. If you turn to the agenda you will find that one resides in Mexico, two in London, one in Sumatra, and one in Central America In other words, there is not a single local man down in the list. I am afraid some of you are getting rather slack in bringing forward new members. Possibly you are content with a membership of 1,000, but I would prefer to see it 2,000, and you should always remember that nomination forms are at your disposal.

DRIVING DEVICE FOR FEEDERS.

Mr. Bevington tells me that he has brought his driving device for feeders here to-night. I will ask him to show it to the meeting.

Mr. R. G. Bevington: You will remember that at the last meeting I referred to a driving device mentioned by Mr. McDermot as having been invented by Mr. Nelson in Western Australia, but the description given was so meagre that I could not quite catch what the actual device was or what it could possibly be. He spoke of it as being so very simple, and according to the ideas I then formed I had a driving device made. Since the last meeting I have received a letter from Messrs. Fraser & Chalmers, informing me that they are the holders of the patent rights of the Nelson driving device, and they also sent me a little booklet describing it, which I found to be absolutely and entirely different to the drive which I had made. The Nelson driving device is worked from the rock shaft of the feeder which does not extend backwards in the ordinary way. On one side it is brought straight down from the centre bracket, is then cranked up again and has a sort of slot at the end of the lever which engages with the edge of the *circular table* of the feeder and not in the *friction disc*. As the rock shaft works, it moves this arm and keeps on turning the feeder table round. That is the Nelson device. This one which I now show you I claim to be quite original. It has been in use continuously for 10 weeks. I will pass it round, and you will see how very little wear there has been on it. The edge of the friction disc goes into the centre slot, and those I am having made now are made solid, in order to give a little more weight. This drive is only hinged at the back end. It simply hangs by its own weight on the friction disc and it gets the upward pull from the rock shaft lever. Naturally the slot gets a grip on the round edge of the friction disc and moves it. You get a very delicate movement and it is a simple device. Those of you who are interested will see really what a light drive it must be if you look at the small amount of enlargement which has taken place in the round hole at the small end of the arm. There is very little wear in the slot, showing the lightness of the drive.

AN IMPROVEMENT IN TIPPING POTS DURING SMELTING.

By W. D'Arcy Lloyd (Member).

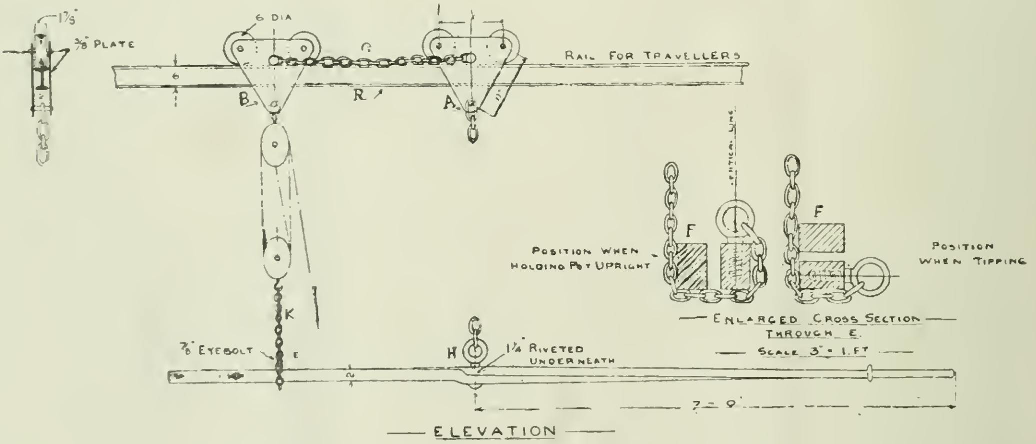
The following procedure I have found so useful that I make that fact my apology for placing it before the Society, thinking that it might prove of advantage to others as it has been to me.

The operation of pouring pots when taken from the reverberatory furnace as worked, appearing to me to be somewhat cumbersome, I set about trying to devise some means whereby it might be done quicker, with less handling of the pots, and in a cleaner manner, and also with less labour.

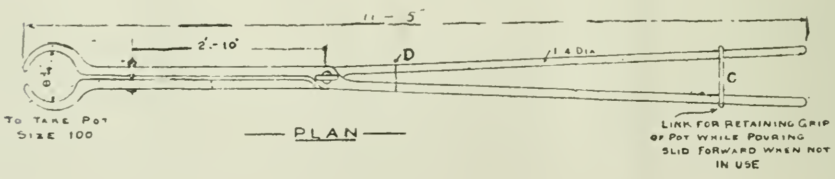
By means of the device shown in the diagram two white men and two natives can conduct the pouring operation quickly and easily.

The device adopted consists of a chain, block and tackle, a short piece of chain about 4 ft. 6 in. long, and an eye-bolt fixed in the tongs. The man in charge of the smelt takes out the pot from the furnace, the grip of which is retained by the sliding link C (see diagram below): the pot is placed against the lip of the mould. The other white man (a learner) pulls on the tackle, and the pot is tipped, delivering its contents steadily into the mould. The tackle is slackened off, the empty pot placed on an iron stool and the link C slid forward, when the tongs are ready for the removal of another pot from the furnace; one boy takes the empty pots from the stool, the other boy operates the furnace door. The moulds are placed on an iron bench, the tops being on a level with the floor of the pot chamber. This method has been worked at our last three smelts at the Cason Gold Mines, Ltd., and is quite a success.

Shiftsmen are not taken from their work to assist at pouring, whereas formerly at least three were required; the operation is quicker with consequently less chance of the furnace cooling off; the slag is also quite free from splashes of gold on the side of the cone. In the diagram R represents an overhead rail, A the carriage that carries the tongs, A and B run on



ELEVATION


11 – 5
2'-10"
D
1¼ DIA
C
TO TAKE POT SIZE 100
LINK FOR RETAINING GRIP OF POT WHILE POURING SLID FORWARD WHEN NOT IN USE

PLAN

the rail R. G is a chain on each side of A and B, being equal to E—H, the chain G preventing A and B from getting farther apart than this distance. C is a link for retaining the grip of pot when pouring. D is the position of the link when taking pots from the reverberatory and allowing freedom of action: E position of attachment of tipping chain. F F shows in section the tongs and position of tipping chain before and after the pot has been tipped. When about to remove pots from the reverberatory the tongs are pushed forward, B also runs forward, butting against and pushing A before it, whilst on the return B drags A by means of the chain G. It will thus be seen that A is always in the correct position for a straight pull on the tongs when they are being tipped. When all pots are poured the chain K can be unhooked at E, and block and tackle can be pulled up out of the way when the fresh pots are being put in.

The President: I move a vote of thanks to Mr. Lloyd for his interesting little note.

I wish also to say that we have received an invitation from the management of *South African Mines Journal* to visit the rock drill trials. I would certainly advise you to take advantage of this invitation. I think everyone must now realise that manufacturers are making strenuous efforts to give the mining industry a drill which will solve our labour problem, and they are certainly making headway in that direction. The present trials are probably on a larger scale than any that have hitherto been attempted, but one must, of course, admit that they are not going to be conclusive, for, unfortunately, they cannot well take into consideration the question of maintenance, nor allow for the fall in efficiency which must necessarily follow as wear and tear takes place. These points are left out of account, but the trials are giving us much valuable information, and it is a unique opportunity for seeing how these one-man drills behave not only on the surface but underground—the latter being, of course, the more important.

NOTES ON TIN MINING IN CAPE COLONY.

By Harry D. Griffiths, A.R.S.M., Wh.Sc. (Member).

During the past few years the failure to locate new and extensive goldfields in South Africa has diverted the energy of the prospector and mining investor into new channels. The occurrence of cassiterite in the Bushveld and of copper in the Zoutpansberg, coupled with a substantial rise in the price of these metals, proved a considerable attraction, and indicated a new outlet for activity. Although the initial results obtained in the Bushveld did not prove entirely satisfactory, they nevertheless had the effect of drawing the attention of the mining engineer to the possibilities of the red granite in that part of the country. This formation was minutely and extensively searched, and this led to further mineral discoveries, some of which have reached the producing stage or are on the point of proving of some importance, thereby confirming the views expressed by a few as to the commercial value of that formation.

The interest evoked in the location of tin is unabated, and a few notes upon the working of one of the most extensive individual alluvial and reef propositions in South Africa may prove seasonable and of general interest and value to the members of this Society.

The Kuils River tin mine is situated at a distance of some 17½ miles south-east of Capetown, on the eastern rising edge of the Cape Flats, and in the angle formed by the main railway line and the Stellenbosch branch. The ground at first rises gently from the west, and then more steeply on reaching the western boundary of the property. A range of hills runs roughly in a north and south direction, the highest point of which attains a height of some 800 ft. above sea level. One main depression or creek runs to the west at right angles to the hills, and it is in this that the cassiterite was originally formed. The view looking from the property is one of great beauty, as nothing intervenes to hide the Cape Peninsula from Simonstown to the Lion's Head.

The property in more peaceful times was typically agricultural, thousands of vines and fruit trees having their roots, so to speak, in cassiterite. But the miner has now superseded the farmer and has begun to unfeelingly destroy those products of the soil, upon which so much patience and care have been lavished, and to replace pastoral serenity, by hustling and muddy activity.

Historical.—Some time before the war, as report has it, some old and experienced prospector mistook the soft decomposed granite for fine grained sandstones, and the oxide of iron scum in the pools for indications of petroleum, and he induced some people to start prospecting for coal and petroleum. Needless to say, that search was unsuccessful, but the farmer pointed out some black heavy granular substance occuring in the creek, and this on analysis proved to be cassiterite. The discovery induced a renewal of activity: trenches were cut on the banks of the creeks and a few shallow pits were sunk. Then, on the high ground quartz outcrops showing cassiterite were located. Two syndicates appear to have been then formed, one to work the portion called

"Annex Langverwacht" for alluvial tin, and the other to exploit the quartz reefs which run from Langverwacht into Hasendal.

The low price of tin about this period, and the subsequent disturbance of business during the war, had the effect of making the work spasmodic and unsustained. Pessimistic views as to the prospects began to find expression when the syndicate's funds ran short, and an attempt at reconstruction appears to have failed for some reason or another. In the middle of 1905 the writer, accompanied by the managing director of the present company, made a lengthy examination of the property, and concluded that it had good prospects as regards alluvial as well as lodes, and that it was worth being seriously tested. A ten months' test was undertaken, and gave such satisfactory results that it led to the formation of the present company. The property of the Kuils River Tin Mines, Ltd., consists of the farms Hasendal, Rosendal and Annex Langverwacht in one continuous block, and Eikeboom and Saxenburg, separated from the former by small portions of intervening land, and totalling some 8,918 acres. The north end of the hill rises somewhat sharply from the southern portion of Hasendal, and cuts across Langverwacht into Saxenburg, throwing off a branch to the west, near the southern boundary of Rosendal. The principal denudation of the hills is therefore to the north-east and west, and it is in those directions that the tin bearing wash or deposit occurs.

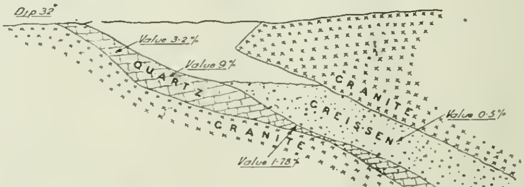
The property is fairly well watered, the portion Hasendal being traversed by the Bottlary Spruit, which may be reckoned to flow at an average rate of about 20,000 gallons per hour. On Langverwacht several springs feed the creek at the rate of about 40,000 gallons per 24 hours. As no work is done at night, the water making during that time is carefully conserved in a series of small dams, the largest one having a capacity of 1,500,000 gallons. The slopes of the hills are covered by surface drift, underneath which the tin bearing wash is obtained; the higher portions are somewhat bare with numerous outcrops of rock.

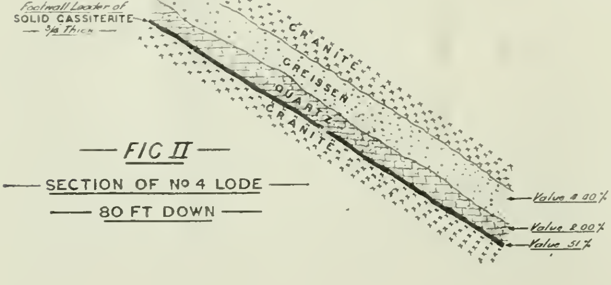
Geological.—The hills are composed of grey granite, similar to that encountered in the Peninsula. It is coarsely porphyritic in structure with a large quantity of biotite, and it forms part of an important mass some 14 miles long and 6 miles wide, intrusive in the Malmesbury schists. Its direction is a little west of north, and makes an acute angle with the strike of the schists. The western junction of the two formations cannot be far distant from the western boundary of the Langverwacht.

Lodes.—Near the top of the hill there are several strong outcrops of quartz, following a north-south direction, and occurring on five fairly distinct lines, with a width between extremes of some 1,500 ft. Some of the outcrops have been opened out by means of trenches and shallow shafts, and show the characteristics of true lodes. The dip is to the east, and flat, seldom exceeding 35°, and each quartz occurrence appears to be accompanied by well defined bands of highly micaceous greisen, mostly in the hanging wall. The walls are of decomposed granite, occasionally showing tourmaline, and display a tendency towards a gneissic structure. The thickness of the quartz is variable, in places pinching to a mere stringer to open out again within a short distance to a width of 3 or 4 ft. The bands of greisen appear more regular and constant, becoming thickest where the quartz is thinnest (Figs. I. and II.). The width between the footwall

— FIG I —

— SECTION OF No 4 LODE AT OUTCROP —





— FIG II —

— SECTION OF No 4 LODE —

— 80 FT DOWN —

of the quartz and hanging wall of the greisen is fairly regular, and in places attains 8 ft. Both quartz and greisen carry cassiterite, the former in large irregularly disseminated masses of a brown or black colour, or in stringers following footwall and joints, the latter being in fine particles evenly disseminated, of a brownish colour showing only on panning. Wolfram also occurs in places where the quartz is poor in cassiterite, the two appearing to interchange with one another: no wolfram has so far been detected in the greisen.

The granite intervening between the lodes has not been explored; but generally its high state of decomposition would suggest the idea that it has been subjected to decomposing agencies, such as ascending vapours or solutions, and that it may form part and parcel of a wide mineralised zone. It is likely therefore that in places it may show masses impregnated with cassiterite similarly to the occurrences in the Potgietersrust district. The existence of large masses of tourmaline included in the decomposed granite supports this view. The flat dip of the lodes is probably only local and due to a subsidence to the west of their top portion. If the same dip had continued in depth some of the lodes would have been found outcropping on the sharper eastern slopes of the hill.

The outcrops can be traced through the property for a distance of about a mile. There are evidences of the presence of one or two more quartz lodes further to the west in closer proximity to the probable junction of the granite and schists, and they would appear to have greisen bands in the footwall instead of the hanging wall. The lodes seem worth being opened out and in places are so rich that with preliminary spalling high grade ore could be sent to the reduction works. The configuration of the hills lends itself to opening them at the outset by means of adits, which would give some 200 ft. of backs.

Alluvial.—The northern and western slopes of the hills are covered with sand and detritus attaining in places to a thickness of 40 ft. This is underlain by a wash or deposit consisting at its lowest portion of a loose breccia of quartz lumps held together by small grains of the same material and a little clayey matter. The wash carries cassiterite mostly in well rounded grains. The thickness of the breccia varies greatly, attaining some 8 ft. but with an average of 4 ft. The variations in thickness are due to the irregularity of the footwall, which is, as a rule, well defined, and consists of highly decomposed granite or pot clay of uneven and undulating surface. The upper portion of the wash consists of a mixture of clay and small grains of quartz, rarely attaining ¼ in. in diameter. This is streaked by bands of clay of variegated colours and in places attains 12 ft. in thickness. It carries tin in payable quantities, but to a less extent than the breccia. This top portion is also quite distinct from the overlying sand or detrital granite. A typical section in the present face would be as follows:—

Black sandy soil 1 ft. carries a little tin
Red ,, 10 ft.
Yellow clayey soil 7 ft. or detrital granite } Overburden.
Top wash 5 ft. carries payable tin
Breccia or tin wash 4 ft. rich in tin.

The rock fragments in the breccia are exclusively composed of white or dark quartz, from angular to rounded in outline. In places they are of large size. Occasionally a few highly rounded boulders of greisen are encountered, but they are rare. In the interstitial gravel a little clay with fine flakes of mica is found, the gravel itself being all quartz. The top lines of the two washes are extremely regular for long distances. It sometimes occurs that the wash seems to have been cut off by granite projections which at one time must have stood as pinnacles, around which the wash is again formed.

The Cassiterite.—All the cassiterite contained in the wash may be considered as coarse, as little of it will pass through a sieve of 36 mesh to the linear inch. The largest piece obtained so far weighs 3½ oz. The following represents the average grading of the cassiterite recovered from the wash:—

Remaining on 6 mesh	...	12·82 per cent.
Through 6 mesh on 8 mesh	...	19·41 ,,
,, 8 ,, 10 ,,	...	12·88 ,,
,, 10 ,, 12 ,,	...	9·25 ,,
,, 12 ,, 20 ,,	...	25·55 ,,
,, 20 ,, 36 ,,	...	20·09 ,,
		100·00

This does not take into account the fine tin which is found in the top portion of the overburden, and which is occasionally treated, nor of that which remains in the very fine slimes. For the purpose of shipment three grades are made up as follows:—

Remaining on 10 mesh	...	No. 1 tin.
Through 10 mesh on 20 mesh	...	No. 2 tin.
,, 20 ,, 36 ,,	...	No. 3 tin.

A typical smelter's return for these grades is:—

No. 1 tin	...	75	per cent. of metallic tin.
No. 2 tin	...	75·25	,, ,,
No. 3 tin	...	63	,, ,,

Impurities.—The impurities accompanying the cassiterite in the wash are occasional, viz., small quantities of wolfram, titaniferous iron and limonite. The wolfram occurs as the wash approaches the line of strike of the lodes, the titaniferous iron occurs in a finely divided state mostly in the top overburden, and the limonite occurs where the wash is only covered by a small thickness of surface soil. The limonite and the ilmenite may therefore be considered as extraneous impurities of more recent origin. Neither of these can be entirely separated from the cassiterite in the dressing of the fine stuff. A recent analysis of the ore shipped gave the following results :—

	No. 1 Tin.	No. 2 Tin.	No. 3 Tin.
	Per cent.	Per cent.	Per cent.
Cassiterite ...	95·84	96·23	80·47
Silica ...	2·65	2·48	5·43
Iron oxides and alumina ...	1·25	1·16	8·62
Titanium oxide	0·12	0 09	5·30

Some of the fines which all passed through 36 mesh, obtained from portions of the wash where it is immediately overlain by several feet of black sandy overburden gave the following analysis :—

Cassiterite ...	16·46	per cent.
Silica	15·34	,,
Iron oxide and alumina	34·13	,,
Titanium oxide ...	34·00	,,

The presence of the silica in the ore is not due to inefficient dressing but is accounted for by the fact that a small proportion af "specimen tin" (quartz with adhering cassiterite) is included.

Origin of the Cassiterite.—In his original report, the writer expressed the view that most of the tin in the wash was derived from the breaking up of the quartz and greisen lodes. This has not been unanimously accepted by some engineers who have visited the property, and who have attributed its origin to the disintegration of the granite mass itself, which they assume must have been impregnated with tin, the main argument being that the lodes, taken generally, appear too poor to have been able to produce such a rich wash. As far as is known by the little work which has been done, no impregnated granite has yet been found. The quartz boulders in the wash are identical with that obtained in the lodes and the numerous pieces of "specimen tin" found, invariably show no other rock than quartz. The argument from the low grade nature of the lodes is not forcible, as one must remember that when the denudation which produced the wash took place, the outcrops must have stood at a greater altitude, and the denudation washed away the decomposed felspars and mica leaving the quartz from the lodes to form the wash. The thickness of the wash does not represent the whole result of the denudation, but its highly concentrated product. So that even with lodes which on the whole would be low grade, a rich wash could be produced.

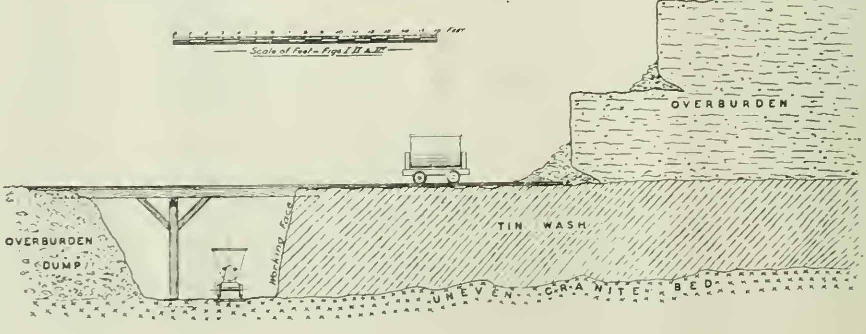
There is no evidence left to show that the granite of Kuils River was at one time overlain by the "Table" sandstones. During three years' work only one piece of sandstone of some 1 lb. in weight was found. On examination, it is quite unlike the Cape sandstone, being very fine-grained with many indurated coloured and parallel streaks, and probably belonging to the Malmesbury series.

The denudation and consequent formation of the tin wash must have taken place after the sea had receded from the portion of the Cape Flats now uniting the Peninsula to the main land. The wash is totally unlike one which would have been formed on a shore line, and no marine fossils have ever been found either in the wash or in the overburden. The granite forming the base of the wash is highly decomposed, but shows distinctly that it was eroded whilst it was still in a comparatively fresh condition, and that the decomposition of its felspar into kaolin took place after the wash had been formed. Its outlines are all highly rounded, smooth, and undulating, showing

FIG III

SECTION THROUGH WORKINGS

SHOWING METHOD OF REMOVING OVERBURDEN



it to have been subjected to considerable and violent attrition.

Working.—At the origin, when work was commenced in the bed of the creek, the lower rich portion of the wash only was found with a very small thickness of surface sand. As the work proceeded in the banks, the thickness of the overburden increased until now, at some points it is 30 ft. thick. The dip of the wash is very regular whereas that of the surface varies considerably. At some points the wash is actually exposed at the surface. In the present main working face the overburden is stripped in benches 8 to 10 ft. in height, and in advance of each other of from 8 to 10 ft., as shown in Fig. III., the benches are undercut from 3 to 4 ft. with vertical cuts 20 ft. apart, and are brought down by means of long iron bars being jumped in from the next bench, at a distance of 4 ft. from the edge. The overburden is then trucked and dumped over the worked out portions of the wash, a clear space being left in front of the wash for the ore tram line.

The wash itself is brought down on the same principle, only one bench being necessary. The undercut and side cuts are not, however, quite so deep. As the stuff comes down easily when damp, each excavating boy is capable of bringing down from 4 to 6 cub. yd. per day. The ore is loaded into 16 cub. ft. trucks by two boys to each truck, and run down an average of 200 yd. to the boxes and dumped. The work done per native under these conditions is as follows:—

Excavating, 12·66 loads of 16 cub. ft. per boy per day.

Filling and tramming, 15·10 loads of 16 cub. ft. per boy per day.

The corresponding figures for the removal of the overburden are not available, as the work is being done on contract, but it can be reckoned that 150 boys bring down and remove some 480 cub. yd. (measured *in situ*) per day, or at the rate of 3·2 yd. per boy.

The removal of the overburden is a costly item, which would have been considerably reduced had hydraulicing methods been feasible. The bringing of the necessary water would involve a considerable capital expenditure, and high cost of pumping, as the nearest available supply would be 5 miles distant, and the water would require to be pumped up 700 ft. But the main and deciding consideration is that the resultant tailings could not be disposed of without seriously encroaching upon adjoining properties, and silting up the Kuils River.

Tin Washing.—Until recently the method of recovering the tin was by box sluicing.

Australian Sluices.—These were erected at the inception of washing operations, and have been in use constantly. They consist of two puddling boxes A and B (Fig. IV.) 30 ft. long each. The larger one A is 2 ft. 9 in. wide at its topmost end, and widens to 4 ft. 6 in. The small one is 2 ft. 6 in. widening to 3 ft. 6 in. They slope at the rate of 1½ in. to the ft. Four feet from the

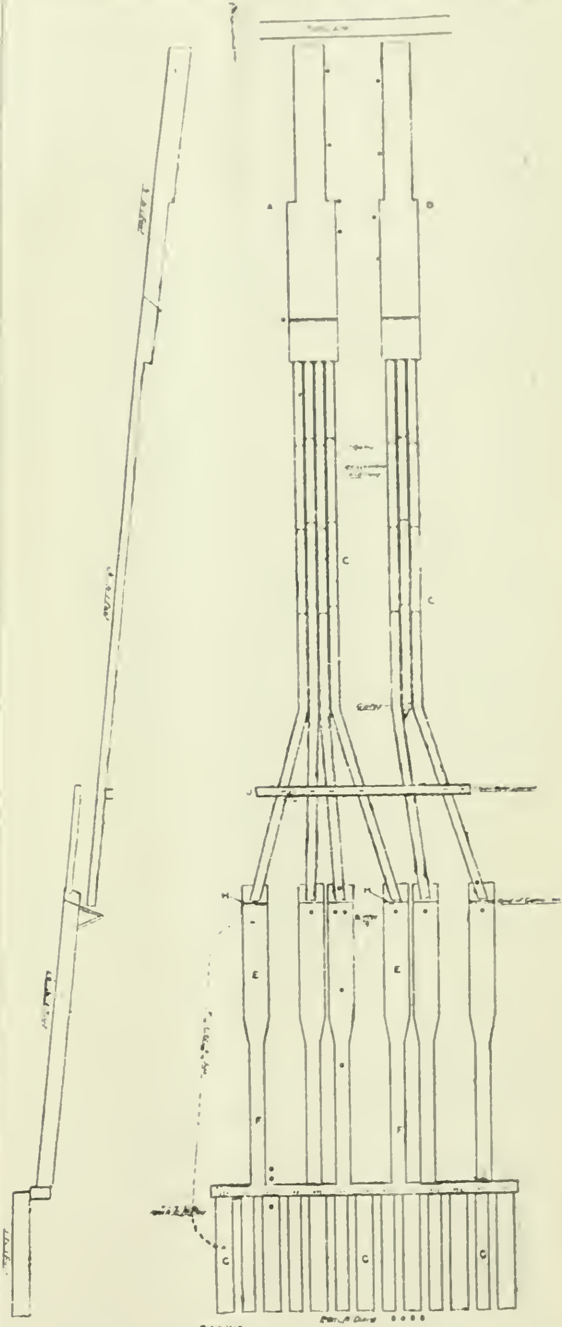
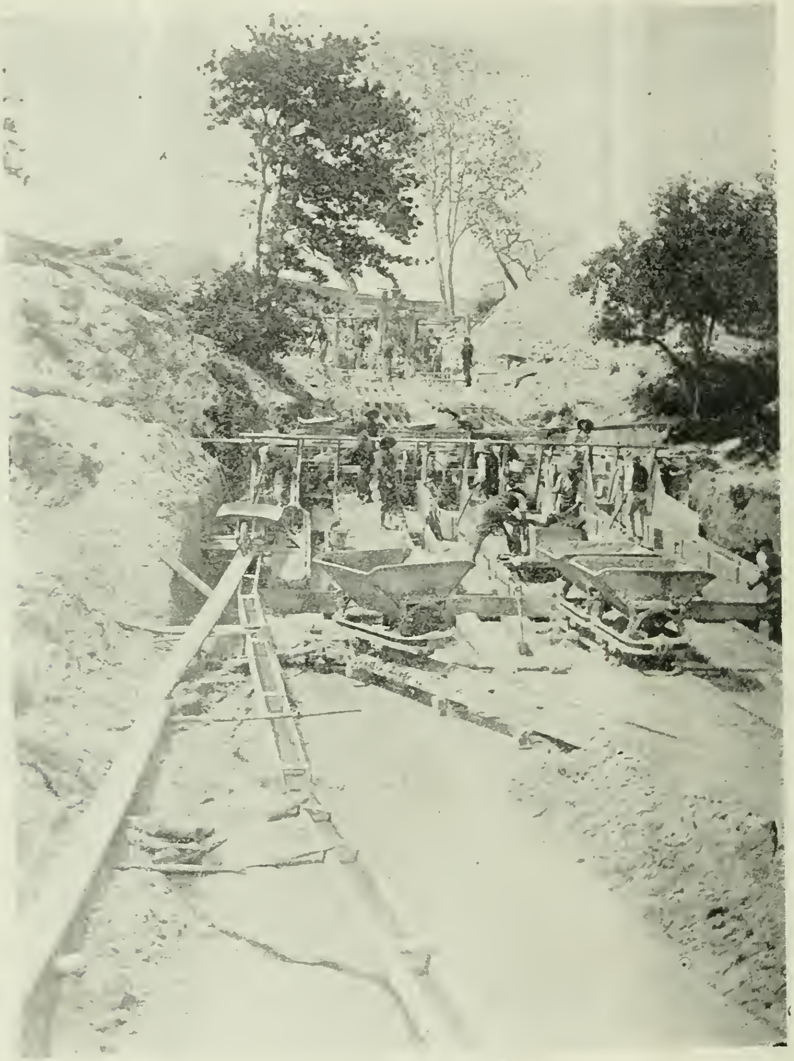


FIG. IV.—AUSTRALIAN SLUICES (TO SCALE).
• Indicates position of men.

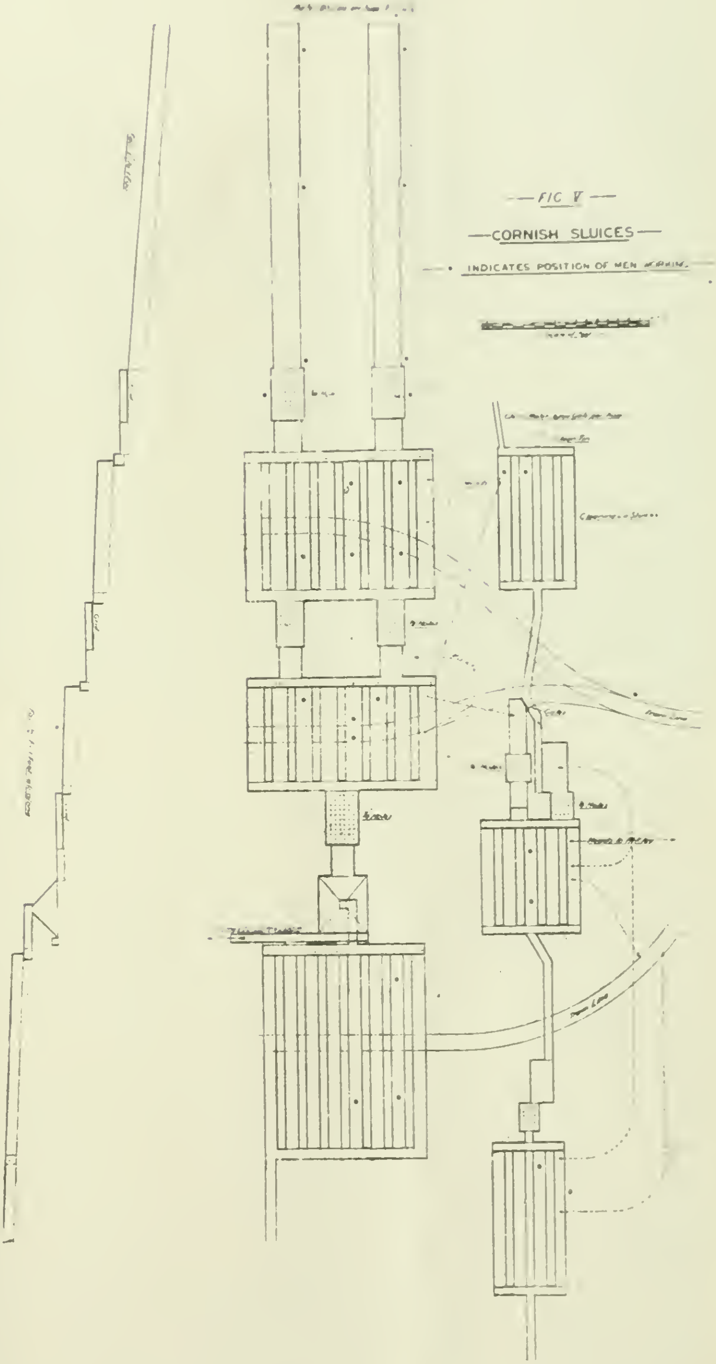


AUSTRALIAN SLUICES.

lower end an incline screen of $\frac{3}{8}$ in. vertical bars placed 1 in. apart extends across the box, and prevents stones and lumps finding their way to the sluicing boxes. Four launders, 9 in. × 9 in. at an inclination of $1\frac{1}{4}$ in. to 1 ft. with 3 gates to each lead from A to 4 sluices E E. From B 3 launders lead to a similar number of sluices. The sluice boxes are 13 ft. 6 in. long, 2 ft. 3 in. wide for a length of 11 ft., then narrowing to 10 in. for the last 2 ft. 6 in. At the top of the sluices are fixed wire gratings H, composed of stout galvanised iron wire set vertically $1\frac{1}{2}$ in. apart and bound together by finer wire every $3\frac{1}{2}$ in. The sluice boxes are continued by a launder 15 in. wide, 18 in. deep (F) leading into a cross launder 12 in. wide × 14 in. deep, provided with holes and plugs for allowing the pulp to fall down the setting pits G, which are 11 ft. long and alternately 18 in. and 13 in. wide. There are two of these pits to each sluice box, and they are provided with gates permitting the gradual concentration of the heavy sands which have escaped the sluice box.

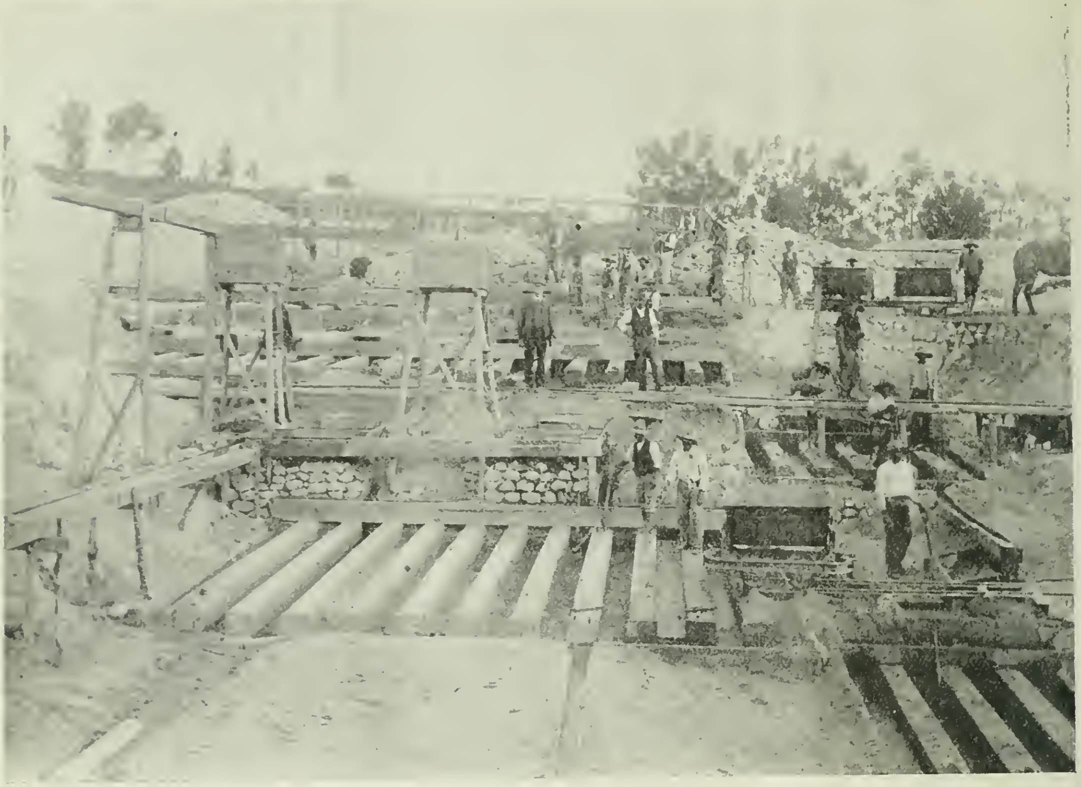
Method of Working.—The stuff being dumped at the upper end of the puddling boxes is met by a strong current of water. Four natives with shovels push the stuff down, at the same time breaking the clay lumps as much as possible. Stones and lumps which have not broken up are then taken out by means of a grid shovel, loaded into barrows and taken to dump. These lumps before being rejected are examined for any adhering cassiterite picked up from the bottom of the box by rolling clay, and the quartz boulders showing cassiterite are put aside. A rough concentration takes place in the launders C by means of regulating gates over which the stuff flows before it reaches the sluice boxes. Here again the lumps retained by the finer screen are taken out to dump. Three boxes are allowed to receive the full flow given by the launders, and the stuff is being constantly moved up against the stream, first, by three boys, and then by one boy. One box is in the meantime running with clear water supplied from launder J, and the cassiterite being cleaned thoroughly at its head. What escapes the constant raking up in the boxes and the following launders settles in the pits G, which are attended by one boy, who also gently rakes up in each pit alternately. The middlings from these pits are sent back to the sluice boxes to be reworked again in conjunction with the other stuff, and the headings or fine sands are taken out for further treatment on the Wilfley tables. On the diagram the position of the boys employed has been indicated. The number of hands required to work the 6 sluice boxes described is 26. The average amount of water used, including clean water for cleaning up, is 776 gallons per load of 16 cub. ft., and the maximum capacity of the sluices is 13·4 loads per hour.

Cornish Sluices.—These are shown on Fig. V. They consist of 2 puddling boxes 2 ft. 10 in. wide for 15 ft., narrowing to 2 ft. 6 in. wide for the remaining 18 ft., and sloping 1 in. per ft. These are followed by iron gratings 3 ft. × 5 ft., set horizontally with $\frac{3}{4}$ in. round holes, from which the lumps are taken off to dump. The two puddling boxes lead by means of a transverse launder with distributing holes to 8 sluice



—FIG V—

—CORNISH SLUICES—



CORNISH SLUICES.

boxes 17½ in. wide × 17½ in. deep, 11 ft. 9 in. long, sloping ½ in. to the ft. These are followed by a cross launder and 2 horizontal gratings, 2 ft. 6 in. × 4 ft. 3 in., having ½ in. round holes leading into 8 more boxes 18 in. wide, 17 in. deep, 8 ft. 9 in. long with a similar slope. A cross launder at the end of these is followed by a grating 3 ft. × 5 ft. wide, ¼ in. round holes, through which the stuff passes into a pointed box which eliminates all the very fine sand or slimes. The coarser sands drawn from the lower portion go to 8 settling pits 13½ in. wide, 12 in. deep, 18 ft. long.

The cleaning up arrangement shown on right hand side of Fig. V, consists of 4 boxes, 13½ in. wide, 11 in. deep, 11 ft. 9 in. long, followed by 2 pairs of inclined boxes and gratings, and 4 more boxes 13½ in. wide × 11 in. × 9 ft. These again lead into another inclined box, grating and 4 settling pits 13 in. × 11 in. × 12 ft. 6 in.

In order to save a lengthy description the *modus operandi* has been carefully indicated on the plan. It will be noticed that in the system the stuff is more minutely manipulated than in the Australian sluices, and that nearly all the sands have to be shovelled out. In the case of the Australian sluices, provided that adequate fall is obtainable at the end of the settling pits, no sands would have to be lifted out and trucked, and from 4 to 5 boys less would be employed.

The cleaning up arrangement in the Cornish installation is the more effective and leaves little chance of losses through carelessness or negligence on the part of those employed. The principal figures in connection with the two installations are given in the following table :—

	Australian Sluices.	Cornish Sluices.
Space required	107′ × 30′	126′ × 31′
Total fall required ...	17′	16′
Slope in puddling boxes ...	1½″ per ft.	1″ per ft.
Slope in sluicing boxes ...	1¼″ ,,	1½″ ,,
Men required	26	27
Capacity loads (16 cub. ft.) per hour	13·4	18·7
Water required per load gal.	776 gals.	597 gals
% of fine tin recovered from sandstreated by Wilfley tables	34·81	18·80

The two methods, therefore, differ in their capacity, quantity of water required and efficiency. The percentage of the fine tin which has to be recovered from the sands by the Wilfley concentrators shows that the Cornish method, owing, no doubt, to smaller inclination, and lesser rush of water, is the better saver of fine tin. The concentrates from the Australian sluices contains less "Nos. 2 and 3" tin than those from the Cornish sluices, and on cleaning up lose 32·02% of their gross weight as against 26·62%.

Similarly, the tin recovered by the Wilfley tables from the sands of the Australian sluices contains more coarse tin than those from the Cornish sluices. Although the manipulation of the stuff is greater in the latter method, the greater efficiency and the greater proportion of tin recovered direct, give it a decided advantage. In a place where water would be an important consideration, the Cornish method would commend itself.

Before Wilfley concentrators were in use, the sands were cleaned up in inclined boxes, but the process was tedious and lengthy, and a comparatively poor concentrate was obtained.

Losses in Sluicing.—Owing, no doubt, to the coarse nature of the cassiterite grains in the ore, the losses in the sands are practically nil, and the tailing dumps would not even repay the most saving of Chinamen. Losses, however, occur in the lumps of ore which have not been broken up thoroughly and ore taken out at the different screens. Lumps of stiff clay which themselves carry no tin, are apt, on rolling over in the boxes, to pick up loose grains of cassiterite on their surface, and although such occurrences are watched, it is obviously impossible to avoid a certain loss. The lumps are dumped separately and in such a way as to be easily secured for further treatment. The lumps of ore, though soft, are tough, and do not disintegrate readily. After immersion in water for 48 hours they can, however, be broken up fine. Exposed to the atmosphere, they weather only after a few months, provided they have been laid out thin. These lumps come mostly from the top portion of the wash which carries only fine cassiterite, and they contain a small amount of clayey matter, and form at the most 5% of the quantity of ground wash. The lumps of pure clay and quartz fragments form another 10%. The value of the ore lumps naturally varies greatly, but they would generally pay for rewashing, provided a cheap and efficient method of doing so were devised. This question is now under consideration. Had water been more plentiful on the property, a jet of water under pressure would have answered the purpose, as has been shown by trial. It is estimated that one-third of the lumps sent to dump would yield from 2 to 3 lb. of tin per ton, or a value at present of from 2s. to 3s.

The sluicing methods described can therefore be considered satisfactory and suitable to the nature of the wash treated, and efficient in their extraction.

The Rotary Pan Method.—Quite recently, washing by means of a rotary pan, similar to that used in diamond washing, has been tried. The experimental pan is 8 ft. in diameter and 9 in. deep. It is fitted with the usual channel iron arms, eight in number, each carrying eight knife-edge teeth set from 4¼ in. to 1⅜ in. apart. The ore is screened through a rotary screen 2 ft 6 in. diameter, 10 ft. long overall and composed of ¼ in. plates with holes from 1 in. to ¼ in. square. The inclination is 1½ in. per ft. The shaft is hollow and fixed, and is perforated with ¼ in. holes, through which water is sprayed inside the screen. The screen is carried by two main cast iron spiders with three arms, and one intermediate with four arms, running on the hollow shaft. The thrust is taken up against a substantial cast iron disc at the lower end. A series of experiments have been conducted by the writer, and it was found that the involute-setting of the teeth as in diamond washing answered best. The teeth are set ¼ in. from the bottom of the pan, and the inclination of the knives to the radial arms requires to be slightly less than in diamond washing. Several speeds of rotation were tried, 10 rev. per min., or a peripheral speed of 245 ft., was obtained. A greater speed could not be attempted, as the pan sides were not deep enough to keep in the swash. At this speed the solid matter near the circumference was 3½ in. deep, whereas the outlet was 3 in. above the bottom. The cassiterite is concentrated within a ring 4 in. wide round the outer edge. The inner sections of the pan on repeated testing showed no cassiterite. The low setting of the teeth effectually prevented the settling of the fine cassiterite over the bottom of the pan.

The great difference in specific gravity between the cassiterite and its accompanying matter makes a higher peripherial speed possible. By doing this also a larger quantity of ore can be treated, without detriment to proper concentration. In the new installation designed by the writer, a speed of 365 ft. is adopted for 12 ft. pans with sides 12 in. clear depth, and it is anticipated that from 500 to 600 loads per ten hours will be treated by each pan.

The results of the experiments with the 8 ft. rotary may be summarised as follows :—

Maximum capacity of pan : 12 loads per hour
Water consumed per load : 81 gal. for sandy ore
137 for clayey ore.
Power required for installation : 6 h.p.
Greatest speed of outside teeth : 245 ft.

The ore is fed by two natives at the rate of 100 to 110 loads per day, and two boys attend to the pan and the removal of screen lumps to dump.

In the new installation the feeding will be done by means of a water jet.

The tailings run by gravitation, and a portion of the muddy water is taken from a screen in the tailings shoot and conveyed by a bucket elevator to the shoot leading into the trommel. The buckets are 12 in. × 10 in. × 6 in., and are elevated at the rate of 32 per minute. The heavy deposit is drawn as required (continuously in the case of rich ore) through a 1¼ in. hole, whose edge is flush with the side of the pan. From ore going 8·20 lb. of cassiterite per ton, 24·57 lb. or 1·09 per cent. of concentrates are obtained. Of this quantity 55 per cent. pass through a "twelve mesh" and yield 40 per cent. of clean cassiterite; 45 per cent. remain on "twelve mesh" and yield 30 per cent. of cassiterite. The results vary slightly, according to the richness of the ore washed, but an average concentrate containing from 35 to 38 per cent. of cassiterite can be reckoned upon with a speed of 245 ft. The man attending to the drawing of the concentrate is now so well used to the work that the value of the concentrates is very regular, no matter what the value of the ore may be. In a larger pan, with greater speed, a higher grade of concentrates should be obtainable.

Advantages of the Rotary Pan.—The advantages of the rotary over the sluicing methods are that a less quantity of water is used and that much less labour is required. In plants treating 1,000 loads per day a comparison of the labour, etc., required would be as follows:—

	Cornish Sluice Boxes.	Rotary Pan
White supervision ...	2	1
No. of natives required ...	151	17
Water required, gals ...	597,000	187,000
Power required	4 mules	40 h.p.

The costs of native labour and power reckoned on present basis at the mine would be:—

Cost of native labour...	£18	17	6	£2	2	6
Cost of power ...	0	16	0	1	13	6
	£19	13	6	£3	16	0

Treatment of Rotary Concentrates.—The rough concentrates as drawn from the rotary are not suitable for shipment, and have to be dressed to a value in tin of as near as possible, 75%, this is being effected as follows:—

The concentrates for the day having been collected in tubs are screened through a series of inclined sieves placed above one another in a frame to which a side motion is given by means of a crank driven from the main shaft. The frame moves on wheels, and has a motion each way of 3 in. Water is sprayed at each screen head for the purpose of washing the sifted material down aprons which underlie the screens. The screened material falls at the head of the next screen which slopes in an opposite direction. At the lower end of each screen is a gutter in which collects each grade separately. The screens used from top to bottom are:—¼-6 in. mesh, 8 mesh, 10 mesh and 12 mesh. The stuff remaining on No. 1 screen is generally sorted by hand on a sheet of iron; that passing through 12 mesh is treated over the Wilfley tables. The four intermediate graded products are passed through a hydraulic separator.

A. D. Wilson's Hydraulic Separator.—This apparatus is based upon the principle of the spitzkasten. It consists (Fig. VI.) of a vessel A of the shape of two cones joined at their base and fitted at the top part with a feeding funnel. Inside this vessel is fixed a similar one, having a pipe B at its lower end, and a side discharge

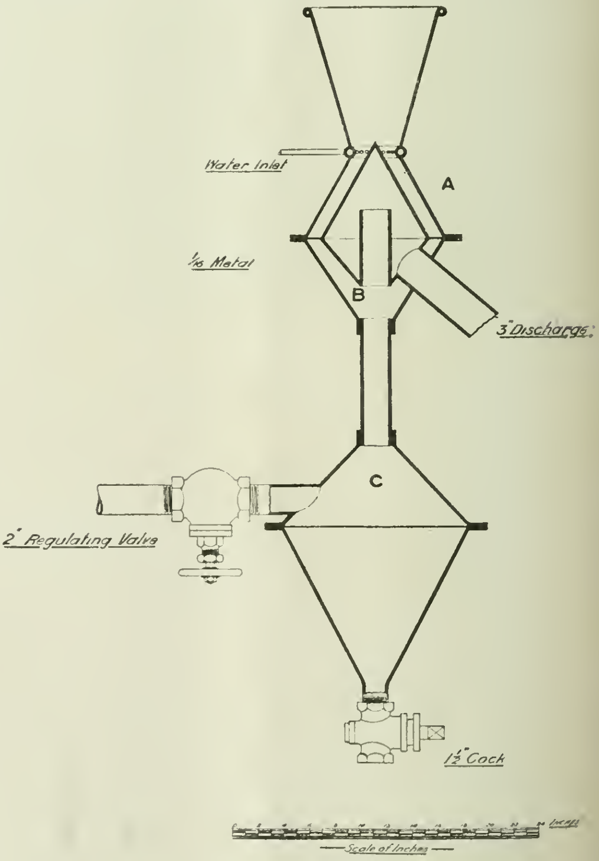


FIG. VI.
A. D. WILSON'S SEPARATOR.

pipe, and of such dimensions as to allow an annular space of 1 in. between the two. The vessel A is connected with a receiving vessel C by means of a 2 in. pipe. The water used for the separation enters the side of the receiving vessel. The upward current meets the ore at the base of pipe B, the lighter particles finding their way up to the discharge pipe, the heavy ore overcoming the current drops into the receiver from where it is drawn out at will through a tap. A valve on inlet water pipe regulates accurately the amount of water required for perfect separation. This, of course, has to be varied for each grade of ore. The results obtained by this apparatus are excellent, the cassiterite being quite clean. The overflow material occasionally contains a little cassiterite which has not found its way into the receiver. This is due to a slight variation in the head of the water, and also to irregular feeding which at the present is done by hand. It is recovered by hand-jigging and repassed through the machine, the hutchings going back to the rotary pan. The value of the concentrates thus obtained is from 72 to 75%. The apparatus is capable of dealing with 5 cwt. of concentrates per hour. The quantity of water used varies from 24 gallons per minute in the case of stuff remaining on 6 mesh, to 10 gallons for stuff passing 10 mesh and remaining on 12. With the same quantity of water and automatic regular feeding, the apparatus would deal with a larger quantity of material. In the new plant 6 of these separators will be used, one for each grade of stuff, and provisions are being made for constant head of water, automatic feeding, and repumping back of water after use. To provide against any irregularity which may occur in the feeding, provision will be made for the tails to be passed over a pair of jigs. The apparatus is simple in construction, costs little, takes no power except that required for repumping back the water, and does not require supervision, lubrication or replacement of parts. It does its work more quickly, and gives in one operation a better product than can be obtained by jigs. Experiments have also been made with concentrates from diamond washing by Mr. Wilson, who claims that he has been able to separate the ilmenite from the garnets. The perfect working of the apparatus depends upon the supply of steady head of water, and careful sizing to obtain equal particles. During the last few months the results of the rotary pan and accessories have been so successful as regards costs, water used and efficiency, that the sluicing method will be entirely superseded.

This is the first time in the history of alluvial tin working that centrifugal force has been employed for concentration, with an apparatus which has been used almost since the discovery of the Kimberley Diamond Fields, and which has, without many improvements, successfully competed in its own sphere with many new methods and principles. It is evident from the experiments at Kuils River, that the greater the difference in specific gravity of the materials to be separated, the easier and closer will the concentration be. A greater degree of concentration than that mentioned could be obtained by allowing the pan to run more "heavy," provided that the teeth are made strong enough. In the case mentioned the results obtained were limited by the speed of rotation, and by the strength of the arms and teeth. The clay present in the ore enabled a thick puddle to be obtained with a back wash lighter than in diamond washing operations.

Losses from Rotary Pan.—The tailings are tested at intervals during the day. For one month they were carefully sampled every hour, and each day's sample was dried, weighed, graded, and carefully vanned. The result was a small amount of titaniferous iron in the grade passing through a 30 mesh, which varied from 0·04% to 0·07%, and which only contained 0·06% of tin, and 1% of monazite. That an infinitesimal loss in tin takes place in the little lumps leaving the pan, there is no doubt. But they are not numerous, and experiments have shown that they would not pay to recover, grind fine, and concentrate. These losses are very much less than those from the screen lumps. For all practical purposes, therefore, the pan losses are a negligible quantity.

Slimes.—A small quantity of very fine or "flour tin" is found in the slimes leaving the sluice boxes. At the outset the quantity was pronounced by Cornish experts sufficient to justify its recovery, and its quality as being capable of concentration. The slimes were therefore conserved in special dams, and a plant was erected for their treatment. This was designed on the old lines favoured by tin streamers. It consists of 3 settling pits, 100 ft. long, 9 ft. wide, and 8 ft. deep, in which the slimes are allowed to settle and the surplus water is eliminated. These are followed by 150 rag frames in 3 double rows, and occupying a space of about 145 ft. × 40 ft. The frames are worked by means of the surplus water from the settling pits, and they are a very old and well-known Cornish contrivance, which is still extensively used for fine tin concentration, and a description would probably be superfluous. The working details of a rag table are given in Fig. VII. The quantity of water required to work each frame is about 95 gallons per hour. The main washing box is allowed to fill and tip in 4 minutes, during which the pulp flows over and gets concentrated and the balance box in 14 sec.

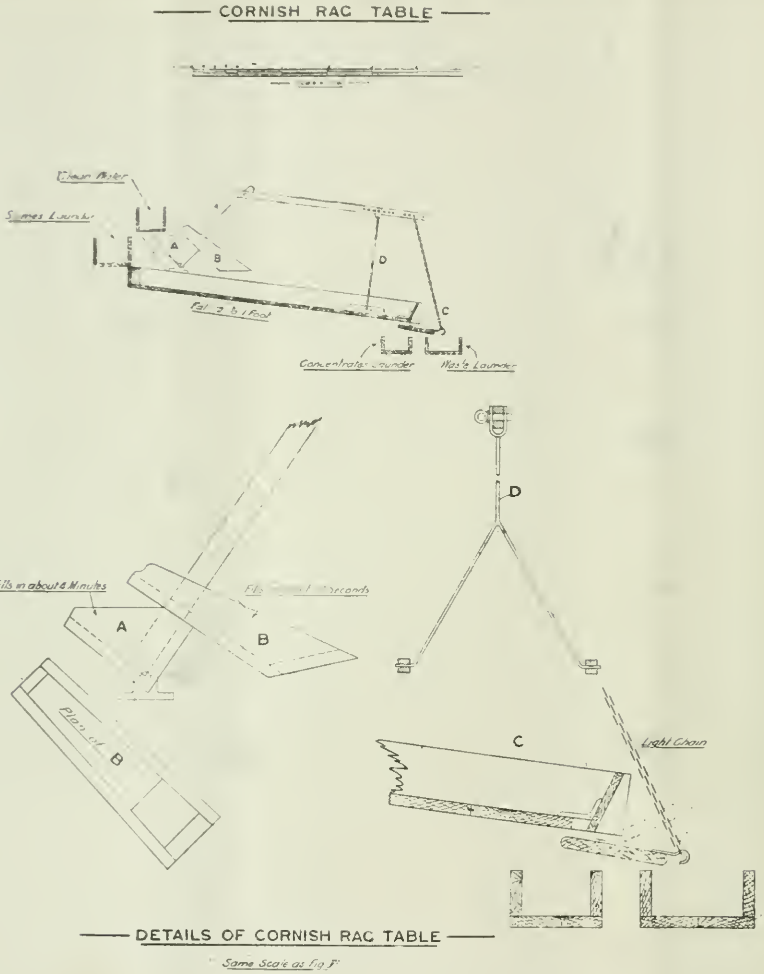


FIG. VII.

These quantities and speeds may however be regulated, according to the quality and quantity of slimes treated.

The rag frames are followed by 5 more settling pits 20 ft. long, 9 ft. wide, and 3 ft. deep, and by another set of 60 frames. From these, and after partial settlement, the slimes pass on to two Acme tables, 20 ft. in diameter. The concentrates are then cleaned up in a concave buddle, 8 ft. in diameter, and finally tossed in kieves. The plant, with the exception of the tables and buddle, is automatic and requires little attendance.

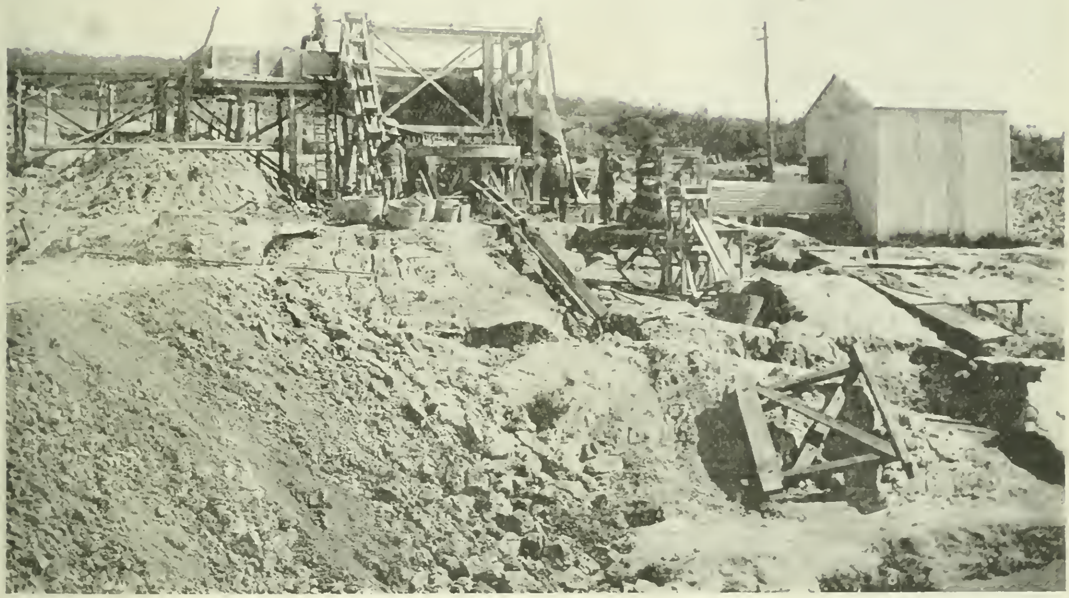
The hands employed are as follows :—

Two natives attending the filling of the different pits and their discharge ; 3 attending the sweeping of the two sets of rag frames, 1 attending the Acme tables, and 1 white supervisor.

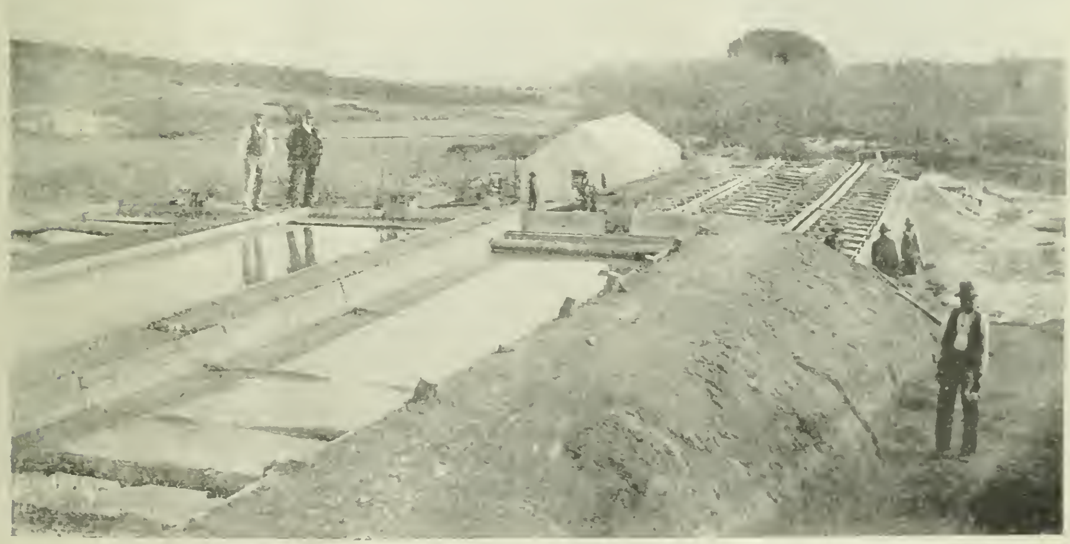
The plant is capable of dealing with 30 tons of slimes per day.

Considerable difficulty has been experienced in obtaining a marketable concentrate. This is due to the presence of titaniferous iron in particles considerably coarser than the accompanying cassiterite. The effect of this on the rag frames and tables is that the coarser iron concentrates to the detriment of the cassiterite. There is no doubt that by grinding, and thereby reducing all particles to the same fineness, the iron will be capable of elimination. This process is now on the point of being adopted.

Power Plant. — All the machinery on the property is worked by electric power, generated by a suction gas engine and producer of the Tangye type, capable of giving 115 b.h.p. The fuel used is Welsh anthracite, costing 65s. per long ton. Working at 50 b.h.p. the consumption of anthracite over a period of 670 working hours has been 31,241 lb., equivalent to 1·07 lb. per h.p. hour or a cost of ·375d. During the same period the



ROTARY PAN.



RAG FRAMES.

cost of attendance, stores, etc., has been as follows:

White attendant ...	0·295d.	per b.h.p. hour.
Native attendant ...	0·099d.	,, ,,
Fuel	0·3725d.	,, ,,
Lubricants and stores	0·080d.	,, ,,
Total ...	0·8545d.	,, ,,

Working at full capacity, it is reckoned that the power could be generated at 0·58d. per h.p. hour, as the attendance and stores would be practically the same.

Yield.—The production of cassiterite at Kuils River, from the inception of washing operations has been as follows:—

Period.	Tons treated	75 ore lb.	Cassiterite per ton lb.
Previous to July, 1905 ...	No record	87,360	...
Aug. 1905-30 June 1906 ...	58,157	288,512	5·00 } 5·714
July '06-1 Jan '07	23,057	107,520	4·66
Jan. '07-30 Sept 07	32,398	253,271	7·81
	113,612	649,303	·255%

Working Costs.—The costs of working during the first nine months of 1907 are shown on the following table. For the sake of comparison the working costs at Mount Bischoff, Tasmania, for 1901 are given. In the case of the latter mine the mining conditions were more favourable, as there was no waste or overburden to remove, and the thickness of the tin bearing formation was very much greater.

KUILS RIVER.	s.	d.
Mining, tramming and sluicing ...	2	6·933
Removing overburden and development and progress work ...	1	7·133
Tin cleaning and bagging ...		1·623
Cost of power and pumping ...		1·520
Cost of working Wilfley tables ...		1·528
Mining stores		2·526
Power and pumping stores ...		1·131
General and office expenditure ...		2·541
Totals	5	0·935

MOUNT BISCHOFF.		
Mining, tramming and hauling ...	3	2·247
Development		2·063
Ore bagging		0·374
Water		0·305
Plant...		3·075
Crushing		9·600
Slimes shed		3·230
Sundries		0·868
Stores		5·226
Management		6·370
Totals	5	9·428

The costs of shipping the ore to London are:—

12 tons consignment

	£	s.	d.
Rail to Capetown	£1	18	0
Dock dues	1	15	0
Forwarding and shipping ...	1	8	0
Freight and primage	9	18	0
B L stamps, etc.		5	0
Agency		7	6
	£15	11	6

£15 11s. 6d. = 26s. per ton

The ruling rates of wages are:—

	£	s.	d.	
Mine foreman... ...	£6	0	0	per week.
Chief sluicer	4	10	0	,,
Rotary attendant ...	2	10	0	,,
Engineer	5	0	0	,,
Electrician	5	0	0	,,
Boss boys } Feed and house		18	0	,,
Boys } themseves		15	0	,,
Pump and engine boys ...	1	1	0	,,
Carpenter	2	10	0	,,
Mason	2	10	0	,,
Blacksmith (General) ...	2	8	0	,,

Two loads of 16 cub. ft. are reckoned as representing 1 long ton. Experiments made with the ore show it to weigh 92 lb. per cub. ft., which after 24 hours sun-drying is reduced to 76 lb. On complete drying the weight is 71 lb. per cub. ft. On the average, 1 cub. yd. of ore *in situ* yields, broken up, 32 cub. ft.

Tin Dressing.—The coarse concentrate from the Wilson separator after being sun-dried are sifted to size and bagged for shipment. Those coming from the sluice boxes are sun-dried on long strips of sacking, sifted into different grades and then cleaned up. In the case of No. 1 and 2 tin this is done during the sifting by agitating the sieve sideways, and gradually getting the light material to the front and throwing it out by a deft jerk. The No. 3 is cleaned by letting it fall from a height upon the level floor. The light particles and some fine tin fall on the outer edges of the heap and are brushed away as sweeping, and thus a concentrate of 75% to 76% of pure tin is obtained. The sweepings are similarly treated, brought to a value of 16 to 50%, and the remaining light stuff is sent to the Wilfley tables and brought to a value of 25 to 40%. The middlings obtained from the operation are poor in tin, but are conserved for regrinding.

The sands from the slimes and the fines (*i.e.*, through 12 mesh) of the rotary are passed over the Wilfley tables, then dried and cleaned as described above, and produce mostly No. 3 tin (75%). The sweepings from all operations are carefully collected, and most of the tin they contain is recovered.

The clean, dry ore is then placed in double bags, the inner one being of strong calico, the outer one of stout canvas, carefully weighed to 56 lb. net, and both bags are sewn up. After stamping the Co.'s name and net weight the ore is ready for shipment. The size of the bags before filling is 13 in. × 11 in., and after filling is about 12 in. × 9 in. × 5½ in at the thickest point. The company justly prides itself upon dressing its tin to such a high value. The cost of labour employed in tin dressing and bagging comes to under 20s., and that of sacking material and sewing to 11s. 8d. per ton of ore shipped. Tin cleaning is exclusively done by Indians whose dexterity is marvellous and almost inimitable.

CONCLUSIONS.

It was anticipated at first that as the workings extended up the slope from the creek the value of the ore would decrease. It is true that in the creek some extremely rich patches were obtained, which led originally to exaggerated estimates of values. But the average has somewhat increased as the work has extended up the slope. The furthest face of work is now some 250 ft. from the creek, and the values are regular and slightly increasing. It is reckoned that so far some 50 tons of cassiterite have been obtained per acre of ground worked out, and that a further 200 acres on Annex Langverwacht alone are tin bearing. This does not represent the whole resources of the company, as the wash has been found at other points.

The work which has been done so far can only be considered as of the prospecting kind, but it is satisfactory to know that the returns are payable and that work on a very much more extensive scale is contemplated, and will be undertaken as soon as necessary financial arrangements are made. The alluvial tin does not appear to be confined to the company's property, as several discoveries in the neighbourhood have been reported, and at least one syndicate is working near the western boundary of Annex Langverwacht, and now producing tin. So far the junction of the granite with the Malmesbury series has not been exposed, and it is not possible to say whether mineralised contact-deposits exist on that line.

The use of rotary pans for tin washing can now be considered as an accomplished fact, and will greatly diminish the cost of labour as well as the consumption of water.

At the end of the paper the author exhibited various specimens of ore from the mine and also many photographic views, showing the various operations, four of which are reproduced, in the text.

The President: It is with very great pleasure that I rise to propose a vote of thanks to Mr. Griffiths. He has long resided in South Africa, and there are very few of our metalliferous deposits here with which he is not familiar. His paper comes to us at a very opportune moment, and I am sure his description of the plant and method of running it will be of assistance to many. What pleases me about his paper most of all is that it is one of the first attempts to give us a detailed description of South African base-metal mining. This has, unfortunately, had many set-backs, but I am glad to see that it is now making progress, and as evidence of this I would point out that our base metal industry in the Transvaal is now employing 197 whites and 2,705 natives, as against 90 whites and 875 natives in 1905, a very marked advance indeed, and the value of the copper, lead, tin, manganese, etc., produced in the Transvaal for 1906 was over £90,000. That shows progress, which I am sure we are all pleased to observe.

Mr. T. Lane Carter: I beg to second the vote of thanks. I think this is one of the most interesting papers we have had before the Society for some time, and that Mr. Griffiths deserves our thanks for taking so much pains in its preparation. The paper will be read not only by those in this room but by people all over the world, and with the greatest amount of interest. The development of the South African base metal-industry is being watched with a great amount of attention, and particularly in America, where they have been trying for years to find payable tin mines, but have not yet succeeded. I note that the method of working is much the same as the method at the Broken Hill mines in Rhodesia, for the mining of zinc and lead ores. I think there is one figure which will strike you forcibly, namely, the cost of 12 tons railage to Capetown, viz., £1 18s. I think that speaks volumes about the economic situation in South Africa. One of the first things required, if we are to have a base-metal industry, is to get cheap railway rates. Another point which fills us with a considerable amount of envy is in regard to the low rate of wages given in Mr. Griffiths' paper. A mine foreman gets £6 a week and workmen £2 10s. I hope they will be able to maintain these figures, but I am very much afraid that if there is to be that expansion in South Africa, which we are all looking for, their wages will have to come up to our level or we shall have to come down to theirs. The future alone will show which it is going to be.

NOTES ON FEEDERS, WITH A DESCRIPTION OF A NEW DRIVING DEVICE.

(*Read at August Meeting, 1907.*)

By D. J. PEPLER (Member).

REPLY TO DISCUSSION.

Mr. D. J. Pepler: Taking first Mr. Beaver's criticism on my paper, I may say that my critic commences by stating that he agrees with but one point in my paper, and that is, that a high stamp duty is very desirable, yet he has failed to recommend any method by which a higher stamp duty could be procured. I am, therefore, not surprised to find that the stamp duty amounts to only 5·1 tons with a 400 screen at the May Consolidated, where Mr. Beaver was battery manager.

After this he continues with a very sarcastic attack on my paper from one end to the other. He commences by giving a description of an old appliance introduced some ten years ago, the operative motion of which is similar to that of a single shaft cart drawn by one horse; the cart is trying for all it is worth to run off the road, the horse again doing all in his power to keep it on the road. The same applies to the feed drive referred to. I know of several mills where it was tried and then rejected. The inventor did not even think its value sufficient to spend protection money on. Again, my critic actually went so far as to accuse me of having "jumped" some one else's idea, because, as he remarked, I am supposed to have seen it working at the Rose Deep some two years ago. When I took charge of the Rose Deep mill a little over two years ago no such drive was in use, though I saw one lying in the smithy in need of repairs. Having known of it for years I paid no attention to it, for I looked upon it as no good for its purpose. Continuing, he claims that the appliance referred to is similar to my device. I contend that there is a vast difference between the two, and perhaps a little more mechanical knowledge will enable my critic to locate the difference.

Again he states that he failed to locate any increase to his stamp duty from the time when he applied these drives to the feeders three years ago. What else could he expect of a defective drive? I did not claim an increased stamp duty by the application of the apparatus in question, but I did and do claim an increased stamp duty by the application of my device to "Challenge" feeders, on account of its reliability in regular feeding, and also a 75% reduction in cost of maintenance. In reply to the conclusion he came to is that the increased duty from the introduction of the new device is due to the fact that my suspension gears must have been in a shockingly bad state, I would mention that at the Ferreira Deep we allow nothing to get into a shocking bad state, everything being kept in thorough good working order. Our stamp duty here is as follows:—1,000 screening, 5·5 tons: 700 screening, 6 tons: 400 screening, 6·5 tons: 300 screening, 7·1 tons; average weight of stamps, 1,150 lb. and 96 drops per minute.

Mr. Beaver also repudiates the correctness of my statement with regard to the reduction of stem breakages since the installation of the new device. This statement was based on actual facts. He claims that badly worn dies and the character of metal in stems have much to do with the life of stems. This, I admit, being common knowledge; but he goes further, and claims that the diameters of pulleys have something to do with the breaking of stems. Now, in my opinion, this latter statement is really not worth commenting on.

He wants to know why I did not compare the new device with some of the older improvements, as for instance, the "Smart" feeder. My answer to this query is, that I do not consider the latter as good as the new device by a long way. A similar combination was introduced by Mr. Tom Watt in 1894. My critic also repudiates the fact that a defective feeder is the cause of the breaking of many cams and cam shafts. Every mill-man with a thorough knowledge of milling recognises that a defective feeder, which allows the mortar frequently to run empty, causes stamps to bang away on bare dies, and strike reverse blows on the cams, is the cause of many a cam and shaft breaking. He also asks why I gave two percentages of less stem breakages with the new device, and doubts my statements. Now, if he will read my paper over again, he will find that I stated clearly, that comparing two batteries running side by side, one feeder driven by suspension gear and the other driven by the new device, the latter showed 50% less stems breaking as compared with the former, over a period of six months, whilst taking the average over the whole mill, the breaking of stems was 40% less in favour of the new device.

In conclusion, my critic asks why if six months was taken up for trial of the new device, and fourteen months have elapsed since, only three-fourths of the old drives have been replaced by the new; and again is of the opinion that if the new device were as good as claimed, the word "hustle" must be unknown there. In reply to this I may state that the makers were slow in forwarding the gear, and the running of the device now over two years shows we take care that we do not hustle upside down but in the right direction.

In reply to Mr. Bevington's remarks I can only say that, having seen the feeder working, he is satisfied with its reliable action, but refers to it as being more complicated than certain other feeders. I certainly do not agree with him in that respect, for, in my opinion, it is simplicity itself, and I should be pleased if he would point out what he considers the complicated parts. In reference to the pawl with the set off in suspension gear mentioned by him, I may state that I have used this also and found it as much of a nuisance as the original.

Mr. Smart states that the new device under discussion is simply an imitation of one introduced by himself in 1899 with a few unnecessary complications. I would like to know if Mr. Smart refers to the so-called "Smart feeder," which I consider quite similar to one introduced by Mr. Tom Watt in 1894, in which case I can only say that I am very much surprised that a gentleman of Mr. Smart's practical experience should fail to see its dissimilarity from mine, which, to my mind, is quite plain to even a tyro, and that by a further study of the sketch accompanying my original paper he may still be able to detect the marked difference between the two gears. In reply to his statement that very little attention can have been paid to the old devices. I can only refer him to the portion of my remarks in reply to Mr. Beaver.

In conclusion, I may say I am prepared to run a test against any other up-to-date device for durability, reliability and cheapness.

NOTES ON THE ESTIMATION OF CAUSTIC LIME.

(Read at August Meeting, 1907.)

By Edw. H. Croghan (Member).

DISCUSSION.

Mr. James Gray: I think we all agree in thanking Mr. Croghan for directing the attention of the analytical members of the Society to the necessity for developing a thoroughly trustworthy method for the estimation of commercial lime. There can be no doubt that it is highly desirable, both from the point of view of economy and of technical perfection in the working of the cyanide process, that the mines should constantly check their lime supplies, and not blindly rely upon the lime-burners' guarantee. The seller learns from experience that his guarantee is accepted in lieu of a test and naturally takes advantage of that state of affairs. I have met with one case where the so-called lime supplied to a mine contained no more than 0·2% of available lime. My experience, in the laboratory with which I am connected, has shown that the quality of the lime supplied to a certain group was subject to very wide variations, until the consignments were regularly tested and the sellers paid on the results, whereupon the variations in quality suddenly assumed insignificant proportions. I believe that to-day only one of our leading mining firms remains unable to recognise the economic and technical advantages of a regular system of testing lime supplies.

Whilst recognising the utility of drawing attention to it, I would like to emphasise more fully the distinction which Mr. Croghan draws between caustic lime and available lime.

I find that the term "caustic lime" has been employed by Andrlik (*Journal Society of Chemical Industry*, 1899), in speaking of what we term "available lime," and that other writers have used the term "free lime" in the same sense. It has been pointed out that to a certain small extent, calcium carbonate, magnesia, and certain calcium silicates, have a protective action on solutions of potassium cyanide, and should therefore be considered to the necessary degree in stating the total "available lime." The term "caustic lime," excludes these substances, and its determination by any method would only be a rough guide to cyaniders in arriving at the value of the lime supplied to them.

The author states that "regarding the existing methods for determining caustic lime, he is not aware of any special literature which has tended to prove or disprove their accuracy." I have carefully referred to the literature on the subject, and I find that the author has overlooked a multitude of references, which tend to show that we are greatly behind the times on the Rand in this branch of analysis. I will not trouble you with the work bearing on the accuracy of the "acid method"—the gravimetric method, by the way, is not even noticed in any recent work—but will confine myself to the history of the determination of caustic lime by solutions of sugar.

Twelve years ago, Stone and Scheuch (*Journal Society of Chemical Industry*, 1895), applied the solubility of lime in sucrose solutions to work out a method of estimating "caustic lime," and I gather from their paper that even they were not the first in the field, though I am unable to trace their method farther back. Weisberg's (*Journal Chemical Society*, 1897) investigations followed in 1897, and Andrlik's in 1899. Andrlik's statements were confirmed by Pellet (*Journal Society of Chemical Industry*, 1900), and later in the same year Pellet and Leduc's investigations were published.

The next work was Maynard's (*Journal Chemical Society*, 1902), and in the following year Bischoff's paper (*Journal Chemical Society*, 1903), appeared. G. W. Williams (this *Journal*, 1905), brought the method before this Society in 1905, crediting its introduction to Crosse, whilst Hendrick (*Analyst*, 1907) and the author have published their work in the current year. This is a brief and very incomplete list of the literature, but quite sufficient, I think, to establish the antiquity of the method.

I will take these papers in chronological order. Stone and Scheuch (1895) found that 150 c.c. of a 10% sucrose solution dissolved 1 gm. of lime and based their method on this result. Weisberg (1897) remarks that it has been proved that sucrose solutions dissolve calcium silicates more readily than pure water, and also, that the amount of salt dissolved increases with the strength of the sucrose solution. He states that the presence of CO_2 increases the solubility of the lime, and also that of the calcium silicates. In the latter case, the silicates are decomposed, yielding $CaCO_3$ and free SiO_2. Andrlik (1899) states that when the limestone is not very pure, calcium silicates and aluminates may occur in the quicklime, and on treatment with sugar solution, they are decomposed, the lime set free going into solution and rendering the estimation false. The quantity of lime extracted, he states, increases with the strength of the sugar solution. Pellet (1900) confirms Andrlik's statements, and points out that water alone is capable of decomposing some of these silicates. Further, the quantity of lime dissolved varies with the nature of the silicate, time of contact, volume of sugar solution and temperature. He also states that the calcium silicates and aluminates may contain more or less iron. Pellet and Leduc later in the same year confirm the decomposition of silicates and aluminates by solutions of sugar, and note that the presence of sucrate of lime in these solutions considerably adds to the extraction of lime from these compounds. Maynard (1902) advises the use of pure glycerol for determining free lime in cements, as it has the advantage over the ordinary sugar method of dissolving lime only. In 1903, Bischoff recommended the use of water instead of 10% sugar solution in estimating caustic lime in basic slags—"as the latter seems also to dissolve calcium hydrogen carbonate, and consequently gives erroneous results." In the light of the above, it would appear that the results obtained by the author with the sugar method are untrustworthy, even for the purpose of determining the caustic portion of the available lime, inasmuch as he has entirely neglected the decomposition of these silicates and aluminates by sucrose solutions.

With regard to Mr. Croghan's theories on the silicates and aluminates existing in quicklime, I find that he assumes the existence of five calcium silicates and one aluminate, to illustrate his calculation by the obsolete gravimetric method.

I have tabulated the various silicates of calcium mentioned by different writers, but I have unfortunately been unable to discover any reference to the body $4CaOSiO_2$ which the author refers to as one of the "well-known" calcium silicates, except in a small metallurgical notebook by H. M. Howe, of New York, where it is supposed to exist in certain slags. Another of the author's "well-known" silicates, $4CaO, 3SiO_2$ appears only as far as I can find in Gmelin's ancient Encyclopaedia and in Howe's Notebook, as a slag constituent. The next silicate I will take is $2CaO3SiO_2$, the existence of which is mentioned by Thorpe and by Mendeléef. Our greatest authority on the subject of calcium silicates, Le Chatelier (*Journal Chemical Society*, 1888) found, however, as far back as 1888, that this body was a mixture of $CaO.SiO_2$ and $2(CaO.)SiO_2$. These two silicates are, according to the latest investigations by Shepherd and Day (*Journal of the Society of Chemical Industry*, 1907), the only definite compounds of calcium oxide and silica, all the other alleged compounds being mixtures of these bodies with each other, or with an excess of lime or of silica.

Mr. Croghan's work in this connection would, therefore, seem to constitute an advance on our present knowledge, and it would be most interesting to have the details of the isolation and properties of the rediscovered compounds, as so far, the only property he has mentioned is their behaviour as regards fusibility in the assay furnace.

In respect to the aluminates, the author assumes the existence of the compound—$3CaO.Al_2O_3$ in the quicklime. I find a number of aluminates mentioned by old and new writers. In the *Journal of the Society of Chemical Industry*, 1882, the following aluminates are mentioned, viz., $CaOAl_2O_3$; $2CaOAl_2O_3$; and $3CaOAl_2O_3$. In 1888, Le Chatelier (*Journal of the Chemical Society*, 1888) prepared and examined the compounds:—$CaO.Al_2O_3$; $3CaO.Al_2O_3$; $3CaO.2Al_2O_3$ (composition not certain). Richardson, Association of Portland Cement Manufacturers, N.J., 1905, examined the following four synthetically prepared aluminates:—$CaO.Al_2O_3$; $3CaO.2Al_2O_3$; $2CaO.Al_2O_3$, $3CaO.Al_2O_3$.

Mr. Croghan has omitted to give us his reasons for selecting tricalcium aluminate as the one aluminate present in commercial lime, nor has he shown that this compound can be formed at the ordinary temperature of the lime kiln: he has even omitted his customary details regarding its fusibility in the assay furnace.

Any value this method may have for the estimation of caustic lime is seriously affected by such apparently slight differences in practice as filtering, decanting or shaking the sugar solution. Mr. Croghan titrates a turbid mixture of calcium and magnesium salts in suspension in a solution of calcium sucrate, a method open to the objection that if the lime is already in solution, the procedure is superfluous and only allows the acid to act on the salts in suspension, thus invalidating the results. Hendrick, on the other hand, titrates a filtered solution of calcium sucrate. The latter method is obviously subject to a small loss of lime, owing to the action of the atmospheric CO_2. There is a third method, that does not appear open to objection, viz., that of decanting the clear solution of sucrate for titration.

To ascertain what difference is given in actual work by these three methods, I have made the following experiments:—

Several quantities of 2 gm. of an ordinary commercial lime were treated with 1,000 c.c. of 2, 5 and 10% sugar solution, and after shaking were allowed to stand overnight. On titrating an aliquot part of the solution after shaking, filtering, and decanting, respectively, the results were as follows:—

	Sugar solution.			
	2%	5%	10%	
Shaking ...	78·68	78·68	78·68	CaO.
Decantation ...	77·56	77·56	77·56	„
Filtering ...	77·28	77·28	77·28	„

It thus appears, as would be anticipated, filtration gives the lowest figures and shaking the highest, varying from 1—2% above those from filtration. The decantation method, which in presence of a sufficiency of sugar to ensure complete dissolution of the free lime appears the most promising process, gives results very slightly above those from filtering. The similar results obtained from the three strengths of sugar solution, show that there is no advantage in using a stronger solution than 2%, *i.e.*, if sufficient liquid be used.

In conclusion, I venture to point out that however stimulating Mr. Croghan's work may be to further research, it brings us no nearer to an exact method of estimating the available or prophylactic lime in the commercial product. He has taken a number of commercial samples of lime of unknown and complex proximate composition—the assumptions based upon the ultimate analyses are largely hypothetical and unsupported by evidence—and he has given us their alkalinity to phenolphthalein when partly dissolved and partly suspended in sugar solution. I would submit that a more scientific method of dealing with the subject would have been to work with mixtures of pure lime, and pure definite compounds of lime, magnesia, silica, alumina, etc., ascertain the proximate composition of the average commercial lime and then apply the results obtained from mixtures of the pure products to the commercial estimation.

I may add that Mr. Croghan's ultimate analyses are not above question. On reference to Hillebrand's investigations (*Chemical News*, 1902) he will find that his method of estimating soluble silica has been proved to be inaccurate.

CALCIUM SILICATES.

Gmelin's Encyclopaedia.	1888 Le Chatelier (1)	1891 Thorpe's Dictionary of Applied Chemistry.	1891 Mendeléef's Chemistry.	1898 Oddo. (2)	1902 H. M. Howe (3)	1905 Richardson (4)	1907 Shepherd and Day (5)	1907 Croghan (6)
$CaOSiO_2$ $2CaOSiO_2$ $4CaO3SiO_2$	$CaOSiO_2$ $2CaOSiO_2$ $3CaOSiO_2$ $3CaO2SiO_2$	$CaOSiO_2$ $2CaOSiO_2$ $2CaO3SiO_2$ $CaO2SiO_2$ $CaO3SiO_2$	$CaOSiO_2$ $2CaO3SiO_2$	Fused 1 mol. SiO_2 with 1 to 6 mols. CaO. In all cases combination only occ'rred in proportions necessary to form orthosilic'tes	$CaOSiO_2$ $2CaOSiO_2$ $3CaOSiO_2$ $2CaO3SiO_2$ $4CaO3SiO_2$ $4CaOSiO_2$	$CaOSiO_2$ $2CaOSiO_2$ $3CaOSiO_2$	$CaOSiO_2$ $2CaOSiO_2$	$CaOSiO_2$ $2CaOSiO_2$ $2CaO3SiO_2$ $4CaO3SiO_2$ $4CaOSiO_2$

1. Le Chatelier, *Journal Chemical Society*, 1888 (abstracts).
2. G. Oddo, *Journal Chemical Society*, 1898 (abstracts).
3. H. M. Howe, Metallurgical Laboratory Notes, 1902.
4. Richardson, Association of Portland Cement Manufacturers, N.J., 1905.
5. Shepherd and Day, *Journal Society of Chemical Industry*, 1907 (abstracts).
6. Croghan, this *Journal*., 1907.

Mr. James Hendrick (contributed)*: So far as I am aware the earliest published method for determining caustic lime by means of a solution of sugar is by Stone and Sheuch. Another possible method which is said to be in use in certain laboratories is by treatment with a solution of sodium carbonate and double titration.

Up to the present time, however, the great majority of analyses of lime are made by determining total lime and certain other constituents, and calculating the amount of caustic lime by a conventional method from these results. As Mr. Croghan has pointed out, such a method is very inaccurate, chiefly because the presence of lime in the form of silicates is ignored. The results obtained by this method frequently make the caustic lime 5% too high. It is most important, therefore, that direct methods of determining caustic lime such as the sugar method, which are both more accurate and far shorter and easier than the indirect method, should be generally adopted. The method of direct titration of the lime sample with standard acid I have found to give fair results, much more accurate than those given by the indirect method, but it also is inclined to give high results, especially in the case of impure samples containing much magnesia and ferric oxide.

My method of applying the sugar method differs somewhat from that of Mr. Williams and Mr. Croghan. It was arrived at after a considerable amount of experiment.

In a 500 c.c. flask is placed 10 c.c. of alcohol. Five grams of the powdered lime are rapidly weighed and added. The flask is filled to the mark with 10% sugar solution, and at once placed on the shaker and shaken for at least four hours. A portion is then rapidly filtered into a 100 c.c. flask, and titrated with standard hydrochloric acid, using methyl orange as indicator. When the residue readily settles clear after the shaking, as is often the case, the solution can be poured off directly into the 100 c.c. flask. In this method a larger amount of lime is taken than in the method used by Messrs. Williams and Croghan. In handling lime it should be exposed to the air as little as possible. If a larger quantity is weighed, the accuracy of weighing need not be so minute as when a smaller quantity is used, and, therefore, the weighing can be quicker. It is easier also to get a really representative sample when a larger quantity is used. My sugar solution is much stronger than that recommended by Mr. Croghan. This has the advantage that a smaller amount of solution is necessary and there is less liability of contamination with CO_2.

(Second Contribution.)

I am disappointed with Dr. Moir's criticism of my paper. Of course, one does not overburden a paper by stating everything, but I assumed that it would be obvious from the context that I knew that a certain amount of sugar is required to form calcium sucrate. He does not appear to have read certain essential parts of the paper carefully or he would have noticed that I do not use a 5% solution of sugar but a 10%, or in other words 10 gm. of sugar per 1 gm. of lime, and that the experiments which he objects to as involving too small a weight of total sugar were carried out with this proportion of sugar and lime. I think, therefore, it is his criticism which falls to the ground.

As to the indicator used I may point out that I did a number of experiments with indicators also, and at first used phenolphthalein, but for a number of reasons gave up phenolphthalein eventually, and now use methyl orange.

It is no criticism of my method to say that it takes no account of magnesium oxide. Dr. Moir will find that I deal with magnesia also in my paper. Further, for agricultural purposes, we do not wish to estimate the magnesia *with* the lime.

As to priority, I pointed out in my paper that the Americans were before any of us in using this method. I read a paper on this subject over two years ago to another Association. (I may say I have been using this method in my laboratory for over five years.) At that time I was not aware that the method was in use in America, and it was pointed out to me by one of the chemists present. In my paper, therefore, to the Society of Public Analysts I embodied references to the word which had been done in America. Your South African work, of course, I had no means of knowing about, as your *Journal* is not usually abstracted for our papers here.

Dr. J. Moir (contributed): I fear that Mr. Hendrick has misunderstood my meaning as regards ratio of sugar to lime; my criticism was only intended to apply to the particular case of Experiment I. of Table III. on p. 125 of this *Journal*, and I do not think even yet that Mr.

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* See also October *Journal*, here only new matter is given—[Ed. Com.]

Hendrick has proved that 5% sugar solution is too weak to use, if one uses plenty of it. As regards the magnesia, I was referring only to the requirements of the Rand, viz., protection of cyanide, and I did not intend my remarks to be a criticism of Mr. Hendrick's method, which is, of course, for *his* purpose the correct one.

I confess that the discussion, especially Mr. Gray's contribution, has opened my eyes on the question of priority, and I regret having claimed for the Rand what it was not entitled to.

EXPERIMENTS IN FIRE ASSAYING AT THE REDJANG LEBONG MINE, SUMATRA.

(*Read at September Meeting, 1907.*)

By G. B. Hogenraad (Associate).

DISCUSSION.

Mr. A. Whitby: I only wish to deal with one point raised by Mr. Hogenraad, viz., the question whether borax is a necessary adjunct in fluxing silicious ores carrying small amounts of other bases or not. I have made a somewhat belated entry into this discussion for reasons which should be apparent, since the main issue was brought before this Society by Mr. White in the discussion on my paper on Routine Assaying. I was under the impression that I had disposed of Mr. White's argument in my reply, but it appears that he is difficult to convince.

That Mr. Hogenraad finds it necessary to scorify his buttons even with his improved flux is surely indicative of imperfect fluxing, because scorification implies the removal of substances which would otherwise interfere with perfect cupellation by carrying some of the noble metals into the cupel.

The fallacy shared by Mr. White and Mr. Hogenraad as to the redundancy of borax in fluxing ores of this description arises from an imperfect conception of the functions of the components of a flux, and also, in Mr. Hogenraad's original flux, from abuse. Excess of borax can be condemned as if none were used, perhaps even more so. The difficulty of forming a true conception of the various duties of the different ingredients in a flux is not made less by the scarcity of data on the subject in our text books, and a really scientific investigation of the matter is badly wanted. If I may be allowed, I should like to quote, however, from a recent work* where the matter of fluxing is dealt with from a somewhat broader point of view than usual, although I do not find much relative to the borates.

Mr. Fulton states that—

"The temperature at which charcoal or argol begins to react on PbO to form Pb is well below 906° C., the melting point of PbO."

"The formation point of a boro silicate $PbO, Na_2O, 4SiO_2 2B_2O_3$, the constituents of which are contained in nearly all assay charges, is 590° C."

"In the fusion of a mixture containing silica various bases and borax glass, that silicate-borate having the lowest formation point will form, and then as the temperature rises absorb either silica or base or both, as these are in excess of the ratio required to form the lowest formation-point compound. If the temperature does not rise high enough to cause this absorption the excess of silica or base or both will remain in suspension."

In another place the author states that silicate of lead is difficultly decomposed by reducing agents, but the borate is easily reduced.

Now, it must not be forgotten that a good deal of the gold is attached to the pyrites: were it free or merely attached to silica, there would be no great use for borax to assist in collecting the gold, but as it is, my former argument is to a large extent confirmed by the author I have quoted. Not only does the borax assist to make the charge more fluid but it also carries the excess of oxide of lead, which is ready to be reduced by the pyrite at the precise moment that this body is being converted into FeO and SO_2 and by the same agency. The soda is not so effective for this purpose and has other duties. But really this is only traversing again the ground covered in my reply to the discussion on my paper already referred to, and since the other side have not brought any arguments beyond Mr. Hogenraad's flux, and the idea of a non-borax flux is scouted by all previous critics, with the exception of Mr. White, I think I need say no more. Soda as a flux is decidedly slow, requires a higher temperature than most of us can get, and is most expensive from its action on the crucible. The same applies to litharge, and I must still maintain that it is possible, if the experimental work is carried out with due regard to all factors, to so assimilate the quantities of borax, soda, litharge and reducing agent as to get a perfect flux which is non-corrosive and gives full values for the noble metals, at the same time being more economical in every way than either Mr. Hogenraad's or certainly Mr. White's later suggestion of a litharge flux. I doubt if this gentleman ever had an average of 8 to 10 fusions out of each crucible with his flux, but it is no uncommon performance with a properly regulated boro-silicate. This, of course, refers to pots of all sizes. The average for some of the smaller sizes is often much better.

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* A Manual of Fire Assaying. Chas. H. Fulton. Hill Publishing Co., London, E.C., 1907.

THE LABORATORY: ITS ECONOMIC VALUE.

(Read at October Meeting, 1907.)

By A. McArthur Johnston, M.A., F.C.S. (Member).

DISCUSSION.

Prof. J. A. Wilkinson: In his opening remarks the author briefly refers to the time when the assayer was looked upon as a "necessary evil," and in a sentence or two bridges the gap which has brought this official into a happier position. The story of this struggle is well known to most of us, and one can only venture at this juncture to express the pious hope that in the case of the testing laboratory this chapter will not have to be recorded. That the post of assayer is even yet not what we should desire to see, is shown from a reply to some questions I put recently to one of them, *à propos* of the use of air in refining cyanide bullion. The title which the author has seen fit to give to his paper is the broad one of "The Laboratory," and with such a title I personally expected a full treatment of the subject. He confines himself, however, to one kind, viz., the testing laboratory, leaving out of consideration the other uses to which laboratories have been and are being put, namely (1) for purposes of instruction, (2) for research. It may be said that the latter often goes hand in hand with the testing of materials. This I am prepared to admit, but, on the other hand, there are few laboratories where this is carried out to any great extent, and in this country I know of none. This kind of laboratory is to be found in factories whose sole object is the production of a special kind of manufactured material from raw products, and the research undertaken has generally for its object the betterment of the manufactured article, the cheapening of the processes of manufacture or the production of new materials which will either displace the old or at least sell just as well in the market. Such laboratories are to be found in Germany, the most brilliant example to my knowledge being the one attached to the Baden Anilin- and Soda-Fabrik at Ludwigshafen am Rhein. About fifteen years ago I was interesting myself in the fitting up of laboratories and had made a final design of a chemical laboratory which I thought would meet all needs, when I had the pleasure of seeing Professor Armstrong's report on the above laboratory. The result was that I had to remodel many of my ideas, as I had never before heard or understood the extent to which a works would go in its efforts to forge ahead. It was in this laboratory that the artificial manufacture of indigo was first worked out, and this result has led to the capture of this trade by this German firm amongst others, and incidentally has almost closed down the indigo fields of India, where the supply was formerly obtained, and which amounted practically to a monopoly. This trade meant thousands of pounds per year to India, most of which has in this method been converted to Germany. This story is but a repetition of the similar one which succeeded the artificial preparation of alizarin by Graebe and Liebermann in 1868, and considering what has been done in this respect, I do not feel it rash to predict that it is only a matter of time—a short time—ere rubber itself, now so largely grown in tropical countries, will be a product of the laboratory. By the researches of Professor Harries our knowledge of its formula and constitution has been considerably advanced, and when once this has been settled, a few years will see its artifical production. The use of a laboratory, however, will appeal much more to a mining community such as this, if we consider for a moment its utility in the manufacture of one of the miner's best tools; I refer to explosives. The chief ingredients of modern explosives are nitro-cotton and nitro-glycerine, both of which, chemically, belong to the same class of substances as those mentioned above. Their discovery and their history tell the same tale, and in consequence the laboratory is the guiding factor in the works. In fact, it would be practically impossible to manufacture high explosives without carefully testing the raw materials used, and also the products at every stage of the manufacture. The fate of a works that tried to economise by dispensing with the laboratory would be that it would sooner or later become a castle in thin air.

The same story can be told in the manufacture of steel, and the advances recently made in this branch have been entirely due to the intelligent use of the laboratory and its resources. The microscopic test used by the author in determining the quality of the shafting supplied is an example of some of the recent work in this direction, this being one of the surest methods of differentiating wrought iron from other kinds of iron or steel. It is, therefore, pertinent to ask, since the value of the laboratory has been amply demonstrated in other countries, how long are you going to buy your lime, your coal, your steel, your cyanide, etc., without a knowledge of what you are purchasing? A comparatively small outlay will put you in possession of that knowledge by the intelligent use of a laboratory properly staffed and equipped. Are you still going to test your boiler water by allowing it to rot your boiler tubes away? Are you still going to test your cyanide by finding that at the end of the month you have used a much larger quantity to get the same extraction? These questions and many others that could be formulated, your own experience will answer. What the author has stated of Germany and America I can corroborate most fully from my own knowledge, and the great progress made during the last two decades by these countries has been built on the solid basis of laboratory work and research.

The particular substances used here and the methods of testing them have been dealt with fairly fully by the author, and the examples given show the good use to which the Goldfields laboratory has been put. In cases where large sums are spent on raw materials during the year the economic value of such a laboratory is placed beyond all doubt, and a true sincere and hearty co-operation between the works and the laboratory will in the end prove the best instrument for economically controlling a mine to the best advantage for all concerned, both owners, employers and employees The interdependence of each upon the other must be fully realised and the sphere of each thoroughly grasped. The laboratory worker must have a clear conception of the functions and limitations of his work, and the practical man should possess a true realisation of the assistance which the other can render. The problems which each has to solve are so different that, unless such a mutual understanding is existent, the assistance which each can render the other will be hampered if not negatived, and the last state will be worse than the first. Both are working for the same object, and it is only by keeping this in view that the maximum success can be achieved.

In connection with that portion of the paper which deals with the methods used in testing the various materials I was surprised to find that the testing of paints had not been carried out. As regards the question of the purity of oxygen when compressed in steel cylinders I was surprised to find that the author had had a belief at some time or other that the oxygen was pure, since it is well known that such is not the case. In one experiment, some years ago, I used the oxygen from this source for a combustion, as it was almost impossible at the time to prepare it myself as I had been accustomed to do, but I found it necessary to pass it through several bottles of KOHAq and H_2SO_4 before I dared to use it, and even then the gas obtained contained about 2% of nitrogen.

Another subject which calls for far more attention than the author has given to it, is the subject of rope greases. This is closely related to the subject previously discussed, namely, oils, but the required tests would be essentially different. It would be interesting to know what has been done in this matter, which in the case of mining is so extremely important. As far as explosives are concerned the tests made by the manufacturers themselves are so good that all that is required in this connection is a check now and then. It must also be remembered here that the Explosives Office of the Mines Department are continually carrying out tests in this direction, and hence the necessity for this on the part of the mines themselves is minimised to a considerable extent.

In conclusion, I should like to add my thanks to those already expressed to the author, and at the same time congratulate the Consolidated Goldfields on their enterprise in establishing such a useful and important testing laboratory, in which I wish them the success, which I am sure they will achieve.

NOTES ON SMALL STOPE DRILLS.

(Read at October Meeting, 1907.)

By E. M. Weston (Associate).

DISCUSSION.

Mr. R. H. Anderson, at the request of the President, then gave a detailed description of a patent bit which was on view, saying that the usual form of starter in practice was $2\frac{1}{2}$ in. to $2\frac{3}{4}$ in. across the cutting edges, while the side or guiding faces of the drill were about $1\frac{1}{4}$ to $1\frac{1}{2}$ in., the cutting edges being forged either by hand or machine. These systems of dressing the bit are not accurate, the principal fault being that each blade is different in length. If there is only a difference of $\frac{1}{16}$ in., one guiding face is sure to be out of action. It is evident that the bit or cutting head should be exact to $\frac{1}{100}$ part of an inch, and that on striking the bottom of the hole each edge should carry its equal share of the shock and each guiding face its share of side pressure. The present drill steel is tapered down to $1\frac{1}{8}$ in. where attached to the chuck, and the piston rod, as a rule, is 2 in.; I have noticed a good many much smaller in diameter—$1\frac{3}{4}$ in. The proportion between $1\frac{1}{8}$ in. and $1\frac{3}{4}$ in. is entirely wrong to transmit the heavy blow necessary for cutting hard rocks. In watching rock drills at work anyone with a mechanical ear will suffer from the harsh sound which is the result of the wrong design of drill steel. The continual jar and shake due to the want of equilibrium causes fractures of the piston rod and the drill steel.

It is left to the care of the blacksmith to gauge the reduction in diameter of the drills, and that is the principal reason why they start the hole $2\frac{3}{4}$ in. and finish up $1\frac{1}{4}$ in. As the rate of drilling is proportional to the volume excavated there would be much saving if you could start the hole with less diameter. With the detachable bit I have invented, the drill steel, is made $1\frac{3}{4}$ in., where attached to the chuck. It is hardened and tapered down to the cutting head. There is just enough guiding face on the drill bit to keep the whole true, and the guiding face wears away without any binding against the side of the hole. The bit is never tight in the hole. The guiding faces are also segments of the same circle, the bit being made in dies, and, therefore, being perfectly accurate. The bit is also exactly true with the piston rod, and it is impossible for the bit to work out of line without pulling the piston rod with it. There is no scoring in starting a hole, and the shock that is delivered through the piston rod is transmitted to the face of the bit and is not absorbed in friction on the side faces or in the loss of energy due to want of equilibrium on the cutting edges. Actual practice with these bits has demonstrated that I can get 100% more efficiency, that is, two holes can be drilled as against one in present practice. I have been fortunate enough to get the May Consolidated to equip their mine throughout with these detachable bits, and they will be in operation within the next four months.

Mr. T. Lane Carter: Can you sharpen these bits?

Mr. R. H. Anderson: No, they are only to be used once.

The meeting then concluded, after the President had expressed the compliments of the season to all present on behalf of the Council.

Obituary.

Mr. Edward Descou Chester (Member), Mining Engineer, formerly resident for some years on the Rand. Mr. Chester was elected a member in July, 1903, and contributed a paper on "A System of Crushing Rock in Stages by Wet Process, and Suggestions as to how this Object can Best be Achieved," which is published in Vol. IV. of the Proceedings.

Mr. W. Roff, formerly a Cyanide Foreman on the East Rand Proprietary Mines, Ltd. Mr. Roff joined the Society as an Associate in Sept., 1897, and was elected a member on the 21st Oct., 1905.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

The Detection of Mercury in Explosives.—"One hundred grams of the substance to be tested is ground up in a mortar with an equal amount of purified French chalk, the grinding being done in small quantities at a time. The mixture is put in a flask in a water oven and attached by glass tubing passing out of the oven to a Woolff's flask or other absorption apparatus containing 50 c.c. of water, with 0·75 c.c. of strong sulphuric acid. Aspiration is applied gently to the whole system, while the flask is heated to the temperature of boiling water. In about two hours' time the mercuric chloride will be transferred to the dilute sulphuric-acid solution in the absorption apparatus.

Electrolysis of the solution with a gold cathode and a platinum anode, using a current of about 0·5 amp. at 2 volts, will yield an amalgam on the cathode which may be weighed and the experiment made quantitative where the quantity of mercury is large. With the small quantities usually present, no decided deposit will be visible on the gold. The gold foil is removed from the electrolyte while the current is still running, washed in distilled water and alcohol, dried, rolled into a cornet and inserted into a small glass sealed tube, 6 mm. in diameter and about 35 mm. long, which has its expanded mouth ground flat to fit snugly against a microscope glass slide. The glass tube is fitted in the hole in a stout brass plate and then very gently heated, after a dry microscope slide has been placed on the top. A sublimate will be obtained on the glass slide, and care must be taken not to raise the temperature sufficiently to re-volatilize this. The slide on being examined under the microscope (about 250 diameters) by transmitted and reflected light will show globules of metallic mercury of large or small size, according to the skill with which the heating has been conducted. If the globules are very small they may not be readily recognised as mercury, but under the conditions of the method any small opaque dots, showing brilliant specks by reflected light, must be mercury, and a repetition of the test will probably show larger globules."—W. A. Hargreaves and W. T. Rowe.—*Engineering and Mining Journal*, Sept. 7, 1907, p. 443. (G. H. S.)

Estimation of Thiosulphate in Presence of Sulphite.—"The reaction is based on the decomposition of thiosulphate by potassium cyanide with the production of sulphite and thiocyanate, the latter being estimated by Volhard's method with $N/10$ silver nitrate solution. The process is carried out as follows:—To an approximately $N/10$ thiosulphate solution are added 1 gm. of potassium cyanide and 2 c.c. of a 15 per cent. solution of sodium hydroxide, both free from chloride, and the mixture heated for half an hour on the water bath. The solution containing sulphite, thiocyanate and excess of cyanide is cooled and made up to 100 c.c. The excess of cyanide is then estimated in 20 c.c. of this solution with $N/10$ silver nitrate by Liebig's process. A known excess of the silver nitrate solution is then added, together with dilute nitric acid, for the complete precipitation of the thiocyanate, and its amount estimated by back titration with standard thiocyanate solution by Volhard's process.—A. Gutmann, *Z. anal. Chem.*, 1907, *8*, 485—500.—*Journal of the Society of Chemical Industry*, Sept. 16., 1907, p. 991. (A. W.)

Corrosion of Iron and Steel.—In studying this problem the authors prepared first a large quantity of pure iron from which alloys were carefully prepared containing known quantities of the different elements occurring in commercial iron and steel. The theories of corrosion are: (1) The carbon dioxide theory first proposed by Dr. Grace Calvert in 1871, and confirmed by Crum Brown in 1888 and G. T. Moody. This is summarised in the equations

$$4(Fe + H_2O + CO_2) = 4FeCO_3 + 4H_2$$
$$4FeCO_3 + 6H_2O + O_2 = 2Fe_2(OH)_6 + 4CO_2$$

(2) The electrolytic theory advanced by Whitney in 1903 founded on Nernst's conception of electrolytic solution pressure and the modern theory of solutions. (3) The hydrogen peroxide theory represented thus

$$Fe + H_2O = FeO + H_2$$
$$H_2 + O_2 = H_2O_2$$
$$2FeO + H_2O_2 = Fe_2O_2(OH)_2 \text{[rust]}$$

The authors first repeated Moody's experiments and got the same results, but they state that his experiments in which the iron was placed in chromic acid are useless because the latter renders the iron passive. Further experiments led them to support Whitney in the statement that iron will dissolve in water which contains not more than a trace of electrolyte, no oxygen, and only that amount of CO_2 which is not retained by a strong alkaline hydroxide.

Further experiments showed:—

a. That the primary function of oxygen in the corrosion of iron is in depolarising those cathodic portions of the iron upon which hydrogen tends to precipitate, and that a secondary function is the oxidation of the ferrous iron ion to the ferric form with its subsequent precipitation as ferric hydroxide.

b. That the rapidity of corrosion in water is a linear function of the partial pressure of the oxygen in the atmosphere above the water.

c. That an electric current can pass between bright iron and iron covered with scale (1) if the concentration of the H ions be increased by the presence of an acid or (2) if oxygen and some other depolarising substance be present.

d. That, in certain instances, areas having marked difference in potential exist in far greater number upon the surface of a piece of iron prone to corrosion than upon iron which is resistant to corrosion.—Walker, Cederholm & Bent.—*Journal Amer. Chem. Society* xxix., 9, 1251, Sept., 1907. (J. A. W.)

Preparation of Hydrogen in the Cold from Iron and Carbonic Acid.—"The reaction which takes place in the spontaneous formation of iron rust: $CO_2 + H_2O + Fe = FeCO_3 + H_2$, may be accelerated by agitation, etc., so as to become a practicable method for the production of hydrogen. In one experiment, 250—300 gm. of fragments of flint were introduced together with water into a steel bottle 50 cm. high and of 2½ litres capacity, and carbon dioxide was passed in until the air was displaced and the liquid was saturated with the gas. The bottle was then closed by a steel cover, and placed in an apparatus where it made about 2,000 revolutions per hour. In another experiment, the contents of the bottle consisted of 10 gm. of cast iron filings, 250 gm. of pieces of cast iron or steel, 250 gm. of water, and carbon dioxide. In both cases there was no appreciable change in the pressure inside the vessel, and after 36—40 hours, the gas withdrawn from the bottle consisted of pure hydrogen. At the end of 20 hours the gas consisted of about two-thirds of hydrogen and one-third of carbon dioxide. If the reaction be effected in a glass vessel, there is a decrease in the pressure within the vessel, owing to absorption of carbon dioxide by alkali from the glass. By introducing nitrobenzene into the vessel together with the other raw materials, aniline can be produced."—A. Bruno, *Bull. Soc. Chim.*, 1907, *1*, 661—662.—*Journal of the Society of Chemical Industry*, Sept. 16, 1907, p. 967. (A. W.)

Use of Zinc in the Assay of Copper Matte.—"The author has substituted zinc for aluminium for removing the copper from solution in the assaying of matte in which iron is also to be determined. Zinc does not interfere with the delicacy of the iodide estimation for copper, so that method may be used. The author finds that with zinc a more complete separation is obtained than with aluminium, the conditions are less rigid and the method is more suitable than the thiosulphate one, and can be advantageously used with most copper-bearing materials met with in works practice, especially when both copper and iron have to be determined. The zinc used is in thin strips, about 3 × 0·25 in., rolled up for about half their length, so that they float vertically in the liquid and extend the whole depth of the solution.

The assay is conducted by dissolving the matte in fuming nitric acid, diluting with 20 c.c. of water, adding 5 c.c. of strong sulphuric acid and heating the solution until white fumes are copiously evolved. When cool, 60 c.c. of water are added, and the solution is heated to a gentle boil when the zinc is introduced and a small funnel inserted in the mouth of the flask. When the precipitation and reduction are complete, the funnel is washed and removed, and a glass wool plug placed in it. The solution is then decanted through the wool, the flask rinsed out with dilute sulphuric acid several times, passing the rinsings also through the wool. The iron in the filtrate is then immediately titrated with permanganate or bichromate. The funnel is replaced in the mouth of the flask, a little nitric acid poured on the wool to dissolve any copper, washing with about 20 c.c. of water. The funnel is removed and the iodide method for copper proceeded with in the usual way. A blank experiment should be tried on each fresh lot of zinc."—D. M. Levy, *Engineering and Mining Journal*, 1907, *84*, 262.—*Journal of the Society of Chemical Industry*, Sept. 16, 1907, p. 970. (A. W.)

Determination of Cyanogen in Mercuric and Complex Cyanides.—"For the determination of cyanogen in mercuric cyanide, the aqueous solution of the latter is made strongly alkaline with sodium hydroxide, and commercial aluminium powder, free from halogens, is added gradually with vigorous agitation. For each gram-molecule of mercury, 12—15 gm.-molecules of sodium hydroxide and 4—5 gm.-molecules of aluminium are used, and the latter is added in the course of 2—3 hours at intervals of 10—15 minutes. When reduction is complete, the solution is filtered, and the cyanogen (now present as sodium cyanide) determined by one of the usual methods. The results are good in presence of chlorine-, bromine-, iodine-, and thiocyanogen ions, and analyses of the compounds, $Hg(CN)NO_3$, $Hg(CN)ClO_4$, and $Hg(CN)Cl$, are described. The method may be used for the determination of cyanogen in complex iron-cyanogen compounds, a weighed quantity being heated with excess of yellow mercuric oxide, and after filtering, treated as described above."—V. Borelli, *Gaz. chim. ital.*, 1907, *37*, I., 429—434.—*Journal of the Society of Chemical Industry*, Sept. 30, 1907, p. 1030. (A. W.)

Reduction of Cerium Dioxide by Hydroquinone and Separation of Cerium from Thorium.—"Quinol (hydroquinone) reduces cerium dioxide to cerous oxide, being itself converted chiefly into quinhydrone, but to a small extent, also, into quinone. For the preparation of cerous sulphate, $Ce_2(SO_4)_3,5H_2O$, a mixture of equal parts by weight of cerium dioxide and quinol, together with distilled water and more than the calculated quantity of sulphuric acid is heated nearly to boiling for 4—5 minutes. Cerous chloride, $2CeCl_3,15H_2O$, can be prepared in an analogous manner from 10 parts of cerium dioxide, 7 parts of quinol, and concentrated hydrochloric acid. The conversion of cerium dioxide into cerous sulphate in presence of dilute sulphuric acid in this way, can be utilised for the separation of cerium from thorium, as thorium dioxide, after ignition, is only slightly soluble in dilute acid."—L. Marino, *Gaz. chim. ital.*, 1907, *37*, I., 51—54.—*Journal of the Society of Chemical Industry*, Sept. 16, 1907, p. 990. (A. W.)

Determination of Chromium in Steel.—"The author gives the following details for carrying out his method of determining chromium in ordinary steels:—Such a quantity is weighed out as will contain 0·02—0·03 gm. of chromium, and is dissolved in sulphuric acid (20% for steels containing 0·5—2·0% of chromium, up to 50% for ferrochromes with 33%), the acid being gently warmed and finally boiled. After the oxidation of the iron by means of nitric acid, or by persulphate, the free acid is neutralised, and then 5 gm. of ammonium persulphate are added. The liquid is then diluted to 400—500 c.c., treated with 20 c.c. of sulphuric acid of sp. gr. 1·18, and boiled for 20—30 minutes. If the sample contain but small percentages of chromium and manganese, it should not be boiled as above, but heated for an hour on the water-bath, and finally boiled for 10 minutes. If manganese peroxide separate, the oxidation of the chromium is complete, but if no manganese be present, the liquid should be cooled to 50° C., 2½ gm. of persulphate added, and the boiling continued for 20 minutes more. Any manganese peroxide is filtered off and washed, and the manganese separately determined; the chromium in the filtrate is determined by addition of excess of ferrous sulphate solution of known strength, and back-titration by permanganate or bichromate.

If more than traces of tungsten be present, the sample is dissolved in 45—50% sulphuric acid, and boiled till no further trace or hydrogen is evolved. Nitric acid of sp. gr. 1·2 is then added gradually, and the liquid boiled till oxidation, both of ferrous salt and of tungsten, is complete. Sodium phosphate (20 c.c. of 10% solution) is then added, to convert the tungsten into soluble phosphotungstate, and the liquid is made distinctly alkaline with a concentrated solution of sodium hydroxide. Dilute sulphuric acid is now added very gradually, with constant stirring, till the precipitated ferric hydroxide is just dissolved. The solution should now be perfectly clear, and the treatment with ammonium persulphate can be proceeded with for the determination of the chromium. The manganese should be determined in a separate portion to which phosphate has not been added."—G. von Knorre, *Stahl u. Eisen*, 1907, *27*, 1251—1256.—*Journal of the Society of Chemical Industry*, Sept. 30, 1907, pp. 1010—11. (A. W.)

Test for Nickel.—"Nickel molybdate is insoluble, in presence of excess of alkali molybdate, in water, or in very weak acid, whilst cobalt molybdate is very soluble. The substance to be tested (containing only cobalt, nickel, or both), is made almost neutral with alkali, and a large excess of saturated aqueous solution of ammonium molybdate is added; the liquid is then heated to 70° C. and shaken. In presence of cobalt, the solution is rose-coloured; if nickel be present, a dense crystalline greenish-white precipitate forms. If the quantity of nickel be very minute, there may be only a turbidity."—E. Pozzi-Escot, *Compt. rend.*, 1907, *145*, 435—436.—*Journal of the Society of Chemical Industry*, Sept. 30, 1907, p. 1030. (A. W.)

METALLURGY.

Preparation of Metal Alloys.—"The object of this invention by Mr. F. Dannert is the preparation of alloys from two pure metals, or an alloy and a pure metal or combination of two alloys, fusion of which was hitherto regarded as impossible, or at least limited to very specific proportions and special conditions. Most varied methods were adopted with a view to solving this problem, but without success. Thus endeavours were made to incorporate oxides with the molten metal, which, it was maintained, were partially absorbed as such by the mass of metal or partially reduced by the said metal. In any case, it was taken for granted that these oxides were eliminated and deposited as metals, when, according to hitherto employed processes, one could not obtain an alloy with the metals composing the primitive metal bath. Quite a different method has since been adopted to free metals from oxides or other impurities by incorporating glass with the metal, and thus forming an intimate mixture by energetic mixing. Glass, thanks to its specific gravity being less than that of the metal, filters, so to speak, through the metal and floats around it, entraining all the foreign substances. Thus a protective coat is formed, above the metal, against the effects of the atmosphere, and running is greatly facilitated. The new process partly avails itself of the advantages which we have mentioned, and also comprises new features. As the difference of electric polarity and specific gravity of the constitutive elements form the chief obstacles to combination of the homogeneous metals, a remedy is found by endeavouring to establish equilibrium between the metallic compounds added and those of the primitive bath. For this the oxides with heavy metallic components are mixed with pulverised glass; or inversely, oxides with light metallic components with heavier pulverised glass The mixture thus obtained is melted and added to the primitive bath. The result is that the metal of the oxide combines with the primitive metallic bath to form a new alloy, perfectly homogeneous in composition, compact, and irrefutably superior to alloys obtained by the old methods. By way of example we can give the following case: Lead oxide is melted with pulverised glass, and to the product thus obtained a kind of frit in certain proportions in a bath of molten copper is added. For example, the following proportions can be adopted: 60 per cent. copper, 40 per cent. lead; or 30 per cent. copper with 70 per cent. lead. Everyone knows the slight affinity of copper for lead, and hitherto it was considered that such an alloy could not be made. With the process just described, however, this alloy can easily be obtained."—*La Métallurgie.*—*London Mining Journal*, Oct. 19, 1907, p. 474. (A. R.)

United States' Mint Charges.—"1. *Melting Charge.*—A charge of $1 (4s. 2d.) shall be imposed for each melt of bullion, except in the case of uncurrent U.S. coin and fine mint bars.

2. *Parting and Refining Bullion* (rate per ounce gross).—On silver bullion containing less than 300 thousandths of gold, three-fouths of one cent (¾d.) per ounce. On bullion containing from 300 to 500 thousandths of gold, two cents (1d.) per ounce. On bullion containing over 500 thousandths of gold, four cents (2d.) per ounce. Silver allowed the depositor shall be calculated on the basis of refining the gold to 990 thousands. When bullion contains less than three-tenths of a thousandth of gold it shall not be reported for the benefit of the depositor. Base bullion containing less than 300 parts of gold per thousand shall pay a charge of two or three cents (1d.—1½d.) per ounce, or such material may be declined, at the discretion of the chief officer. If the alloy be all good copper when but one of the precious metals is present, the charge may be one-half cent per ounce.

3. *Refining Silver* (rate per ounce gross).—A refining charge of three-tenths of a cent (·15d.) per ounce shall be imposed upon silver bullion free from gold, assaying from 998 to 998½, and a charge of four-tenths of a cent (·2d.) upon such bullion assaying from 980 to 997½. Bullion below 980 shall be subjected to the parting or refining charge of three-fourths of a cent (·375d.) per ounce.

4. *Toughening Charge.*—
Gold bullion, ¼ to 2 cents (⅛d.—1d.) per ounce gross.
Silver bullion, ¼ to ½ cent (⅛d.—¼d.) per ounce gross.
This charge is made to cover the cost of fitting brittle fine bars for coinage operations and shall not be imposed upon unparted bars except in special cases where it is necessary to toughen an unparted bar returned to the depositor; or, when the bullion contains iridosmine, etc.

5. *Copper Alloy.*—Two cents (1d.) per ounce of copper required.

6. *Bar Charge.*—On bullion deposited for fine bars not required to be parted or refined, and for standard or unparted bars.

Bars of fine gold per $100 value (£20) ...	$0.04 (2d.)
Bars of fine silver per ounce	.00⅛ ($\frac{1}{16}$d.)
Bars of standard gold per $100 value (£20)	.10 (5d.)
Bars of standard silver per ounce gross ...	.00½ (¼d.)
Bars of unparted bullion per ounce gross	.00¼ (⅛d.)
Small gold bars (less than $5,000) (£1,000)	·05 per $100 (2½d. per £20)
Small silver bars (less than 125 ounces)...	.00¼ per ounce (⅛d.)

—(W. A. C.)

MINING.

Some Recent Investigations on the Hygiene of Subaqueous and Subterranean Work.—"Under the above title an address was delivered, on September 26, at the general meeting of the International Congress of Hygiene at Berlin, by Dr. J. S. Haldane, F.R.S. The first subject dealt with was work under water in compressed air, with special reference to the difficulties and dangers of deep diving, where men are exposed to heavy pressures. It had been found that one of the main difficulties of deep diving was the fact that the divers were very quickly exhausted by any exertion after reaching the bottom, and sometimes became unconscious. The experiments showed that this was due, not to the pressure of the water, as was hitherto supposed, but to an excessive partial pressure of CO_2 in the air of the diving helmet, which is more or less vitiated by respiration. The effects of CO_2 in air depend not simply on its percentage, but on its partial pressure. Thus at ordinary atmospheric pressure (1 atmosphere) 3% of CO_2 in the air would only produce a slight effect; but at 2 atmospheres' pressure it would produce as much effect as 6% at ordinary pressure—namely, violent panting; and loss of consciousness at about 3 atmospheres' pressure or a little more. For this reason the air supply required by a diver increases in direct proportion to the pressure he is under. It was found that the distress and inability to work disappeared when the air supply was increased in such proportion as to prevent the partial pressure of CO_2 from ever rising above 3% of an atmosphere. The depth reached in these experiments was the greatest hitherto definitely recorded—namely, 35 fathoms (corresponding to 93 lb. pressure).

The chief danger of work in compressed air is that from "caisson disease." It was shown experimentally by the French physiologist, Paul Bert, about thirty years ago, that the various symptoms of caisson disease are all due to the disengagement in the blood, and other parts of the body, of bubbles of nitrogen. The blood and tissues become saturated with nitrogen in the compressed air, and the gas is given off as bubbles on decompression. By limiting the duration of exposure and making the decompression slow enough, however, the danger can be avoided. In recent times various regulations have been proposed as to safe durations of exposure and rates of decompression, but very little was known definitely. The matter was therefore investigated experimentally in the steel chamber at the Lister Institute. The results showed that the existing regulations were inadequate, and at the same time furnished clear evidence in favour of a new method of safely decompressing divers and other workers in compressed air. The pressure is reduced at once to half the absolute pressure, as it was found that such a reduction is perfectly safe. Further decompression is carried out by graduated stages. Thus a man working at 45 lb. per square inch of excess pressure, or 60 lb. of absolute pressure (100 ft. of water), is brought up at once to 15lb of excess pressure (30 lb. of absolute pressure, or 33 ft. of water), the rest of the decompression being in graduated stages. By this method a great saving of time and increase in safety is effected. A table was drawn up by the committee, and has been in use for some time in the Navy, showing the rate of decompression calculated to be safe after exposure to different pressures up to 90 lb. per square inch, and for different periods of time. The time-honoured regulation that a diver should descend very slowly was also abolished, as the slow descent adds to the danger and wastes much time. The decompression table is applicable to work of all kinds in compressed air, including the construction of tunnels or shafts in water-bearing strata.

The second part of the address dealt with the effects on men of the high air temperatures met with in mining and other work. Experiments were described proving that no matter how high the actual air temperature may be, it is the temperature indicated by the wet bulb thermometer which matters to a man. If the wet bulb temperature in still air exceeds 88° to 90° F. the body temperature begins to rise, even in a person stripped to the waist and doing no work; and this rise continues until complete disablement occurs. During muscular work the critical wet bulb temperature is considerably lower, and at temperatures of over 80° only very moderate work of a continuous character is possible in still air. On the other hand, movement of the air raises the critical wet bulb temperature by several degrees, and is thus in itself of great advantage to miners working in hot places.

With increase of the wet bulb temperature above 80 F. the working efficiency of a miner gradually diminishes almost to a vanishing point. The mine owner suffers severely in pocket, though there is no definite evidence that the miner suffers in health.

In conclusion, some examples were given illustrating the importance of definitely determining and dealing with the causes of ill-health or loss of working efficiency in connection with various occupations. In diving work excess of CO_2 in the air breathed is often actually harmful: but in connection with mining work injurious effects or loss of efficiency, which detailed investigation proved to be due to such causes as ankylostomiasis or inhalation of stone dust, or excessive heat and moisture, have been wrongly attributed to excess of CO_2 or to other gaseous impurities associated with it, and useless measures taken accordingly. In England a popular superstition seems indeed to have grown up that any air is wholesome if the percentage of CO_2 in it is low enough. The English Factory Act, in the provisions dealing with weaving and spinning factories, where warm and humidified air is used, expressly permits wet bulb temperature of over 90° in rooms where women are working in their ordinary clothes, provided that the proportion of CO_2 does not exceed ·09 %. To take another instance, in the tunnel at present being constructed under the Thames at Rotherhithe, the London County Council, regardless of the enormous expense, have stipulated that the compressed air in which the men work shall at no place contain more than ·08 % of CO_2. This regulation is supposed to be a safeguard against caisson disease."—Dr. J. S. HALDANE.—*London Mining Journal*, Oct. 5, 1907, p. 418. (A. R.)

THE EDWARD MEDAL.—"The *London Gazette* contains the intelligence that his Majesty the King has been pleased to issue a warrant under his Majesty's royal sign manual to the following effect:

Edward, R. and I.

Edward the Seventh, by the Grace of God of the United Kingdom of Great Britain and Ireland and of the British Dominions beyond the Seas King, Defender of the Faith, Emperor of India, to all to whom these presents shall come, greeting!

Whereas we are desirous of distinguishing by some mark of our royal favour the many heroic acts performed by miners and quarrymen and others who endanger their own lives in saving or endeavouring to save the lives of others from perils in mines or quarries within our dominions and in territories under our protection or jurisdiction, we do by these presents for us, our heirs and successors institute and create a new medal to be awarded for such acts of gallantry:—

Firstly.—It is ordained that the medal shall be of two classes which shall be designated and styled 'The Edward Medal of the First Class' and 'The Edward Medal of the Second Class.'

Secondly.—It is ordained that the Edward medal of the first class shall consist of a circular medal of silver with our effigy on the obverse, and on the reverse a design representing the rescue of a miner with the inscription 'for courage.'

Thirdly.—It is ordained that the Edward medal of the second class shall consist of a circular medal of bronze of a similar design.

Fourthly.—It is ordained that the medals shall only be awarded to those of our faithful subjects and others who, in saving or endeavouring to save the lives of others from perils in mines and quarries within our dominions and in territories under our protection or jurisdiction, have endangered their own lives, and that such award shall be made only on a recommendation to us by our principal Secretary of State for the Home Department."—*Colliery Guardian*, Oct. 25, 1907, p. 779. (A. R.)

MISCELLANEOUS.

NEW METHODS OF EFFECTING BOILER REPAIRS.—"On the occasion of the visit of the Institute of Marine Engineers to the Machinery Exhibition, at Olympia, Mr. H. Rucke Keene delivered a lecture on 'New Methods of Effecting Boiler Repairs.' The new methods dealt with by the lecturer were the oxy-acetylene and electric processes of welding. In the first-named process the apparatus consists of two steel cylinders, one containing oxygen gas and the other dissolved acetylene, both under pressure, a special blow pipe, flexible tubes for transmitting the gases from the cylinders to the blow pipe, and small bars or rods of iron or mild steel about $\frac{3}{16}$ in. in diameter, which are fused and which attach themselves to the parts to be united. The oxygen and acetylene gases are ignited at the nozzle of the blow pipe, the resultant flame giving out an intense heat. In welding operations the portion of the plate to be repaired is heated to a welding heat by means of the flame from the blow pipe, the iron or steel bar being meantime held in this flame until a small portion at the end of the bar is melted off and attached to the part to be repaired, the process being continued until the plate is brought up to the required thickness. After welding, the surrounding part of the plate is heated to counteract strains which might be set up by the intense local heat during the repairing process. In the electric process the cable from one pole of the dynamo is connected to some part of the boiler, and the cable from the other connected to the welding bar. This welding bar is fixed to an insulated holder, and an electric arc is formed, which rapidly heats the parts in close proximity to it. At the same time the end of the bar is heated to nearly a molten condition, and the repair is then effected. Samples showing the character of the work done by these processes were afterwards handed round for inspection."—*Times Engineering Supplement*, Oct. 9, 1907. (J. A. W.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

TABLES OF MINERALS—INCLUDING THE USES OF MINERALS AND STATISTICS OF THE DOMESTIC PRODUCTION. By SAMUEL LEWIS PENFIELD, M.A., LL.D. 8vo. VI., X. 88 pp. Cloth. Price $1.00. (New York: John Wiley & Sons, 1907.)

"Part I. of this new book consists of a list of minerals arranged according to the six systems of crystallisation. Part II. gives an arrangement of the minerals according to the elements, with their uses. Statistics of production are also included. In Part III. a number of useful minerals are described, such as corundum, mica and talc. Part IV. gives lists of minerals according to geological occurrence and association. The general classification is igneous, sedimentary, metamorphic and accessory minerals. Veins and vein minerals and minerals resulting from contact metamorphism are touched upon.

For the mining engineer whose geological knowledge is in need of repair this little book will prove most useful. The several methods of classification are suggestive. While by no means a text-book, the volume will be found a convenient and ready source of reference."—*Canadian Mining Journal*, Oct. 15, 1907, p. 465. (A. R.)

INVENTIONS, PATENTS AND DESIGNS—WITH NOTES AND THE FULL TEXT OF THE NEW BRITISH PATENTS AND DESIGNS ACT, 1907. By G. CROYDON MARKS, M.P. Price 3s. 6d. (London: The Technical Publishing Co.)

"The Liberal member for Launceston, in spite of 'the immense success that has attended the unfettered production of the engineering, mechanical, and general manufacturing and constructive industries of Great Britain in face of the open competition of the world,' yet deplores that 'some British manufacturers have a peculiar habit of keeping the nature of their business a secret in the districts where they reside and within the social circle they cultivate;' while in 'America those who are engaged in manufacturing and commercial enterprises are personally proud, and publicly and privately interested in what they are producing.' To remedy this, Mr. Croydon Marks has produced the work before us, in which he explains the meaning of 'patent,' what is patentable, and how patents are obtained. The questions of how to work a patent, the remedies for infringement, and the law relating to designs are explained in the 50 pp. of introduction clearly and with sufficient brevity. Then the statute of 1907 is set out verbatim, with the addition of a very complete table of contents."—*London Mining Journal*, Nov. 2, 1907, p. 535. (A. R.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 453/07. R. Freygang. Improvements in apparatus for washing or extracting minerals. 25.10.07.

(C.) 454/07. F. L. Bergen (1), R. H. Pascoe (2). Improvements in and relating to elevators. 25.10.07.

(P.) 455/07. E. Jarrard. Improvements in and relating to heating and lighting. 29.10.07.

(C.) 456/07. A. E. Sherman (1), J. B. Linley (2). The Improved Chilling Co., Ltd. (3). Improvements in and apparatus for the preservative treatment of food. 29.10.07.

(P.) 457/07. V. N. Halston. Improvements in safety catches for mine skips, cages, hoists and the like. 29.10.07.

(P.) 458/07. W. C. Marshall. Bucket carrier. 29.10.07.

(P.) 459/07. J. Inglis (1), W. T. Hughes (2). Improvements in automatic tipping apparatus for mine trucks and the like. 30.10.07.

(P.) 460/07. J. McGregor. Improvements in safety brake mechanism for mine cages and skips, lifts, elevators and the like. 30.10.07.

(C.) 462/07. A. Gillies. An improved method of and means for pulsating teat cups. 5.11.07.

(P.) 463/07. S. F. Stokes. Improvements relating to bending tests of materials. 5.11.07.

(P.) 464/07. L. Watkins (1), F. J. Warren (2). Improvements in the treatment of the auriferous zinc slimes deposited in the precipitation boxes employed in the process of recovering the precious metals from their ores. 6.11.07.

(P.) 465/07. T. Speedy. Improvements in means for moving the deposit on mine dumps and the like. 7.11.07.

(P.) 466/07. W. Cullen (1), E. H. Weiskopf (2), The British South African Explosives Co., Ltd. (3). Improvements in nitro-glycerine explosives. 7.11.07.

(C.) 467/07. E. M. Wade. Filters. 8.11.07.

(P.) 468/07. A. Pearson (1), W. G. Treadwell (2). Improved inside friction drive for mill feeders and the like. 8.11.07.

(P.) 470/07. F. V. V. Swanton. Improved detachable cutting bits and holders thereof for percussion rock drills. 12.11.07.

(P.) 471/07. M. J. van Lingen (1), S. A. van Lingen (2). An improved apparatus for deep mine pumping. 14.11.07.

(P.) 472/07. L. Pryce. Improvements in means for feeding mills. 15.11.07.

(P.) 474/07. South African Maganite Explosives Syndicate, Ltd. (1), H. C. L. Bloxam (2). Improved method of manufacturing the explosive maganite. 16.11.07.

(P.) 475/07. J. Hart, trading as J. Hart & Co. Improvements in acetylene gas generators. 16.11.07.

(P.) 476/07. G. C. Cullum. Disinfectant powder. 20.11.07.

(P.) 477/07. T. Speedy. Improvements in and relating to the fixing of tappets to the stems of stamp mills. 21.11.07.

(P.) 478/07. B. E. Tennant. Improvements in assaying furnaces. 21.11.07.

(P.) 479/07. F. Robinson. Improvements relating to the conveyance and handling of drills or other articles in and about mines. 21.11.07.

(P.) 481/07. J. H. Hughes (1), W. M. Epton (2). An improved means of the application of springs. 23.11.07.

(P.) 482/07. J. H. Hughes (1), R. H. Harries (2). An improved machine for the pulverisation of diamond bearing material and metalliferous ores. 23.11.07.

(P.) 483/07. J. H. Hughes (1), R. H. Harries (2). An improved machine for washing diamond bearing material. 25.11.07.

(P.) 484/07. F. M. Cox. An improved air lock caisson and appliances for the recovery of precious stones and metals in deep water. 25.11.07.

(P.) 485/07. J. Greig. Improvements in repairing tools, such as rock drills. 27.11.07.

(P.) 486/07. W. G. Williams. Improvements in extracting metals from their ores. 28.11.07.

(P.) 487/07. A. R. Ellis. Improvements in means or devices for use in brushing wagon, vehicle or other wheels; also applicable for trimming felloes and rims of wheels. 29.11.07.

(C.) 488/07. Dr. A. R. Frank (1), M. Voigt (2). Process and device for manufacturing nitrogen compounds. 29.11.07.

(C.) 489/07. C. C. Richards. A combined stamp and grinding mill. 29.11.07.

(C.) 491/07. H. M. Leslie An improved automatic filtering machine. 30.11.07.

(P.) 492/07. B. Cromartie. Safety dynamite cartridge case. 2.12.07.

(P.) 493/07. H. Parnham. Concentrated gold saving machine. 3.12.07.

(P.) 497/07. F. Gewyne (1), A. Roberts (2). Natural position hygienic latrine. 4.12.07.

(P.) 498/07. B. Raffael. An improved wire rope dressing. 4.12.07.

(C.) 499/07. O. H. Fairchild. An automatically operating rotary filter. 6.12.07.

(P.) 500/07. G. Hauft. A new and improved method of pumping water, other fluids or semifluids, more especially such as tailings and other material containing solid matter of different specific gravity. 6.12.07.

(C.) 501/07. E. S. G. Rees. Improvements in or relating to centrifugal pumps and turbines. 6.12.07.

(C.) 502/07. E. S. G. Rees. Improvements in centrifugal pumps, turbines or compressors. 6.10.07.

(C.) 503/07. H. Sanderson (1), J. Isaacson (2), H. St. J. Sanderson (3), E. Wardle (4), J. E. Firth (5), C. E. Charlesworth (6). Improvements in valve gears for steam and other fluid pressure engines. 6.12.07.

(C.) 504/07. A. G. Brandram. Improvements in joints for sewer and like pumps. 6.12.07.

(P.) 506/07. C. G. Lund (1), C. F. Lund (2). Mechanical locust destroyer. 10.12.07.

(P.) 507/07. A. C. Plowden (1), G. F. Jennings (2.) Improvements in rock drilling machines. 11.12.07.

(P.) 508/07. J. Slorach. Improvements in tool chucks for percussive machines. 11.12.07.

(P.) 509/07. J. Welsh. Safety catch mechanism. 11.12.07.

(C.) 510/07. H. C. Behr. Improvements appertaining to means for dewatering crushed ore products or the like, and for delivering the dewatered product into treatment tanks, vats or the like. 12.12.07.

(C.) 515/07. C. J. Grist. Improvements in the treatment of metals such as steel or steel alloys, in order to vary, improve or accentuate the qualities or character thereof. 13.12.07.

(P.) 516/07. W. K. Betty. Improvements in slimes treatment. 13.12.07.

(C.) 517/07. C. Nicosia. Improved artificial stone composition and the process of manufacture thereof. 13.12.07.

(P.) 518/07. W. Daly. Safety catch for vertical shafts with wood or steel guides. 17.12.07.

(P.) 519/07. B. E. Tennant (1), H. Hay (2). An improved process for the precipitation and recovery of gold from leaching solutions. 18.12.07.

(C.) 521/07. N. V. Schoop. A process for autogenously soldering or welding aluminium and alloys rich in aluminium. 21.12.07.

(P.) 522/07. South African Maganite Explosives Syndicate, Ltd. (1), H. C. L. Bloxam (2). Improved method of manufacturing certain explosives. 21.12.07.

(P.) 523/07. C. Garrood. Improvements in hose couplings. 23.12.07.

(C.) 524/07. M. B. Praed. An improved process of carburetting air and apparatus therefor. 23.12.07.

(P.) 525/07. G. C. E. Keet. Improved automatic hydraulic power. 23.12.07.

(C.) 526/07. L. H. Rodgers (1), A. Myers (2). An improved composition for preventing leakage of air through punctures in pneumatic tyres. 24.12.07.

(C.) 527/07. The New Transvaal Chemical Co., Ltd. (1), P. Destefani (2). Improvements in and relating to means or devices for burning paraffin, wax or the like. 24.12.07.

(P.) 528/07. R. H. Anderson. Improvements in percussive rock drills. 24.12.07.

(P.) 529/07. H. Stadler. Improvements in fine grinding machinery. 24.12.07.

(P.) 530/07. K. G. Lund. Mechanical locust destroyer. 27/12/07.

(P.) 531/07. H. L. Burgess (1), W. Philip (2). Improvements in means for automatically discharging liquids from vessels from low pressures. 27.12.07.

(P.) 532/07. A. J. Smith. Improvements relating to the lubrication of axle boxes and the like. 28.12.07.

(C.) 533/07. Duisburgen Maschinenbau Aktien-Gesellschaft Vermolo Bechem & Keetman (1), H. Alfona (2), O. Eigen (3). Improvements in clamping devices for hoisting ropes. 28.12.07.

(P.) 536/07. W. J. Dunnachie. Process for the manufacture and recovery of ammonia and other bye-products and for improving the quality of the tar in the gases of blast and other furnaces, retorts, coke ovens and the like. 28.12.07.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

BLELOCH, W., *to* P. O. Box 3692, Johannesburg.

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COGLE, C. T., *l/o* Fordsburg ; P. O. Box 98, Langlaagte.

COMRIE, A., *l/o* Bulawayo ; Matabele Sheba Mine, Essexvale, Rhodesia.

CRONIN, J., *l/o* East Rand ; P. O. Box 4800, Johannesburg.

CURTIS, A. H., *l/o* Kent ; 48, Castle Road, Bedford, England.

DUDGEON, W. F. H., *l/o* Blinkwater ; Consolidated Goldfields of S.A., Ltd., P. O. Box 67, Bulawayo.

EVANS, A. W., *to* P. O. Box 5492, Johannesburg.

HOWELL, L. C., *l/o* Pietersburg ; 4A, Calle Armenta y Lopez 23, Oaxaca, Mexico.

LUCKE, P. K., *l/o* Saltillo ; Apartado 320, Monterrey, Mexico.

PARRAMORE, W. A., *l/o* Rosedale ; 40, Chomley St., Windsor, Melbourne, Victoria.

REDMAYNE, Prof. R. A. S., *to* 44, Farquhar Road, Edgbaston, Birmingham.

RUSDEN, A. G., *to* P. O. Box 1021, Johannesburg.

RUSDEN, H, *to* P. O. Box 1021, Johannesburg.

RUSSELL, W., *l/o* Germiston ; Nourse Mines, Ltd., P. O. Box 32, Denver.

SEARLE, BARRY, *l/o* Montrose ; Creighton, Cherokee Co., Georgia, U.S.A.

TONKIN, J. L., *l/o* Cornwall ; Mysore G. M. Co., Ltd., Marikuppam, S. India.

WAGNER, P. A., *l/o* Berlin ; Mineralogical Institute, Heidelberg, Germany, until April, 1908, after then to *c/o* Anglo-American Club, Freiberg in Saxony.

WARDEN, P. S., *to* Boston Mine, Battlefields Siding Rhodesia.

THE JOURNAL

OF THE

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OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

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Vol. VIII. JANUARY, 1908. No. 7.

Proceedings

AT

Ordinary General Meeting, January 18, 1908.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, January 18th, Prof. J. Yates (President), in the chair. There were also present :—

48 Members : Dr. J. Moir, Messrs. T. L. Carter, R. G. Bevington, W. R. Dowling, K. L. Graham, A. Heymann, A. Richardson, G. O. Smart, H A. White, Prof. J. A. Wilkinson, W. A. Caldecott, G. Andreoli, W. Beaver, W. K. Betty, A. A. Coaton, M. H. Coombe, E. H. Croghan, G. Goodwin, J. Gray, H. D. Griffiths, J. A. Jones, J. Kennedy, M. Knight, G. A. Lawson, H. Lea, J. Lea, C. W. Lee, R. Lindsay, W. P. O. Macqueen, J. P. McKeown, J. McLennan, G. Melvill, J. E. Metcalf, J. T. Milligan, P. T. Morrisby, T. T. Nichol, W. J. R. North, J. F. Pyles, W. H. Roe, S. Shlom, J. J. R. Smythe, H. Taylor, J. A. Taylor, A. D. Viney, H. Warren, J. Watson and J. K. Wilson.

19 Associates and Students : Messrs. S. J. Cameron, J. Chilton, J. Cronin, W. J. N. Dunnachie, A. L. Edwards, J. H. Harris, G. F. Jones, R. W. Leng, A. G. Rusden, H. Rusden, C. B. Simpson, A. Thomas, A. M. Thomas, W. E. Thorpe, W. Waters, J. Whitehouse, L. J. Wilmoth, H. C. F. Bell and W. H. Johnston.

10 Visitors and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as published in the *Journal*, were confirmed.

NEW MEMBERS.

Messrs. Macqueen and Nichol were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that the candidates for membership had been duly elected, as follows :—

Lathrop, Percy Dunbar, Langlaagte Deep, Ltd., P. O. Box 5, Fordsburg. Surveyor.

Morkel, Arthur, Joker Mine, Abercorn District, *via* Salisbury, Rhodesia. Mine Owner.

The Secretary announced that the following gentlemen had been admitted as Associates by the Council since the last general meeting.

Bell, Willoughby George, B.Sc., Cosmopolitan Proprietary Gold Mine, Kookynie, West Australia. Metallurgist.

Brunton, Charles, Theta G. M. Co., Ltd., Queque, Rhodesia. Cyanider.

Bullock, Lawrence N. B., Butters' Copala Mines, Copala, Estado de Sinaloa, Mexico. Metallurgist.

Moncrieff, Roger Murray, Oriental Consolidated Mining Co., Unsan, Korea. Mining Engineer.

Mosenthal, Bertie, Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte. Assayer.

General Business.

The President : Mr. Hay, the General Manager of the Witwatersrand Deep, has been kind enough to extend an invitation to us to visit the property, and the date has been fixed for Saturday afternoon next, the 25th inst. Quite recently Mr. Salkinson, of that property, gave us a very interesting paper on "The Utilisation of Waste Heat in Slimes Settlement," and it is partly to inspect the particular apparatus connected with this, and also to look at the surface plant generally, which, I may say, is well worthy of inspection, that we purpose making this visit. I would certainly advise all of you who can get away to avail yourselves of the opportunity of seeing the Witwatersrand Deep.

Mr. E. H. Croghan : I would like to draw the attention of members to the present issue of the *Journal*, and to the very clear photographs contained therein. I think the Council is to be congratulated on the excellent manner in which the *Journal* is now produced.

The President : I am sure that will give much pleasure to the Editorial and Publications Committees, who devote a great deal of valuable time to the production of the *Journal*.

We have a note by Mr. G. W. Williams which Mr. Rowland will read to you.

The Secretary: Mr. Williams writes from the Ida H. Gold Mining Company in Western Australia as follows:—

NOTE ON A MODIFICATION OF THE NORMAL TYPE OF BATTERY FRAME.

By Gerard W. Williams, A.I.M.M., F.C.S.

For the past three months the screen frame described below has given every satisfaction, and its adoption has materially decreased the time occupied in changing screens.

The frame is made in 2 in. clear pine in the usual manner. The edges of the screen panels are rabbeted out, ½ in. deep and ¾ in. wide. In this recess an iron frame, constructed of ⅝ in. square rod, fits loosely. Five lugs or buttons keep each frame in position. These lugs are held down by ½ in. bolts, the ends of which are riveted through 1½ × 1½ in. squares of ⅛ in. plate. This plate is countersunk and held in place by screws as shown in the diagram.

The screens are bent on a bench. A block of hard wood, the size of the outer portion of the frame, is fastened to the bench and an iron frame fits loosely over this. The screen is placed on the block, the frame hammered over it and the edges trimmed off.

When a screen in the battery breaks, the lugs are loosened and the iron frame removed. One of the new screens is then put in place, the iron frame replaced and the lug screws tightened. The whole operation only occupies a few minutes. When only one panel breaks, and usually one panel will be found to break more frequently than the other, only the broken panel is removed. A certain saving in the screens is thus effected.

The value of this device is much enhanced when inside amalgamation is used. Changing the panels does not affect the sands which tend to bank up against the lower edge of the screen.

I think that millmen will find this device well worthy of trial, the slight increase in cost of the screen is soon redeemed on the saving of time and material effected.

The President: I move a vote of thanks to Mr. Williams for his little note, and we will now proceed to Mr. J. S. Curtis's paper on "The Origin of Gold in the Banket."

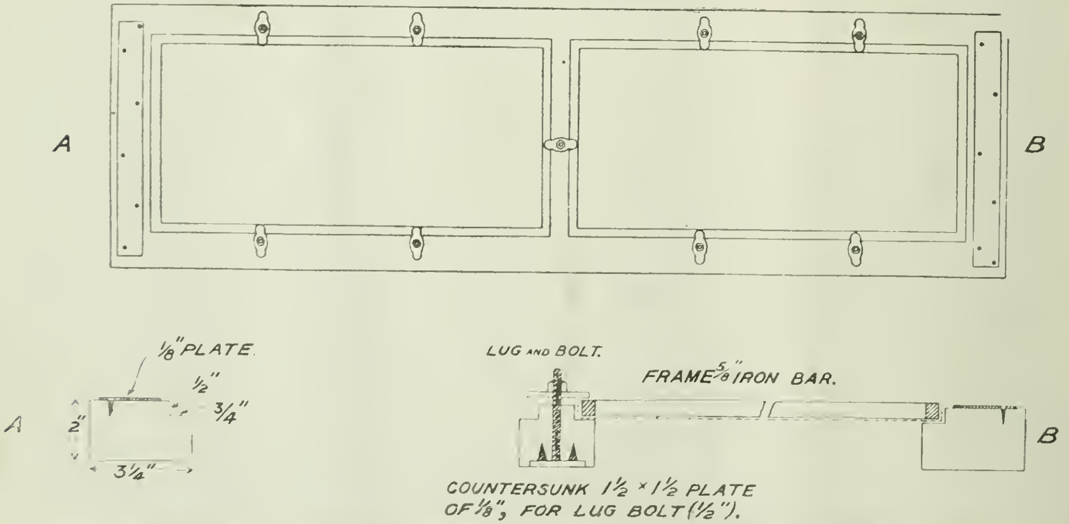
THE ORIGIN OF THE GOLD IN BANKET.

(Having reference to Prof. J. W. Gregory's paper on the same subject, read before the Institution of Mining and Metallurgy.)

By J. S. Curtis (Member).

Professor Gregory is to be congratulated upon the care and research which have enabled him to put in such clear form the numerous theories which have been advanced with regard to the origin of the gold in the banket of the Rand.

Although the object of this answer is to combat some of the views Prof. Gregory has endorsed, and although I cannot agree with him in his opinion as to the origin of the gold, I fully recognise the arguments which have led him to his present belief.



PLAN AND SECTION OF BATTERY SCREEN FRAME IDA H. G. M. CO., LTD., WESTERN AUSTRALIA.

Prof. Gregory's "Summary of Conclusions" contains almost all the points which require answering, so it is probably best to begin at the end of his paper.

I take his divisions and sub-divisions in the order in which they appear.

I. "The theory as to the origin of the banket in best agreement with the facts appears to be that which regards the banket as a marine placer in which gold and black sand (magnetite with some titaniferous iron) were laid down in a series of shore deposits. The gold was in minute particles, and it was concentrated by the wash to and fro of the tide, sweeping away the light sand and silt, while the gold collected in the sheltered places between the larger pebbles. The black sand deposited with gold has been converted into pyrites, and at the same time the gold was dissolved and re-deposited *in situ*."

The theory of the banket deposits being originally marine placers may or may not appear to be the one "in best agreement with the facts"; it all depends upon the point of view. There are certainly a number of people who do not think that theory is a satisfactory solution of the occurrence.

"The gold was in minute particles, etc"—The deposition of the Witwatersrand beds was probably not later than the Silurian, and at that time the rainfall and consequent freshets which transported the sand and pebbles from a considerable distance to the sea shore, was very much greater than it is at the present day, and would certainly have washed away gold of the extreme fineness of that which is present in the banket of the Rand. Fine gold is certainly not characteristic of placers. There is also nothing to prove that black sand accompanied the quartz detritus which formed these beds. The bluish colour of the quartzite of the Witwatersrand beds is due to very finely divided pyrites. Did the fine quartz sand which furnished the material for this rock also contain black sand, and if so, why did it remain with the quartz sand, and why did it not sink to join the black sand in the banket? The earliest evidence of solfataric action is that which was subsequent to the great eruption of basic igneous rock which tilted up the Witwatersrand beds; and the eruption was a very long period after the deposition of these beds.

II. "The distribution of the gold agrees with that of placer deposits in the following respects:

(*a*) The gold as in a placer is contained in the cement and not in the pebbles."

This is true in the main, but it is not universally the case. It is no uncommon thing to find a pebble which has been cracked and the cracks filled with gold bearing pyrites. No better proof of the contemporaneous deposition of the gold and the pyrites can be found, and if the two were contemporaneous it can hardly be doubted that the pyrites brought the gold.

(*b*) "The gold has a wide-spread horizontal and narrow vertical range."

The meaning of this passage is not entirely clear to me. The banket reefs dip at various angles. Some of them have an inclination of 85° (from the horizontal) at the surface, and some of them an inclination of less than 30°. In depth the reefs naturally get flatter. Of course, the gold bearing deposits which are mostly the banket reefs are confined between compact foot and hanging walls of quartzite. Possibly this is what Prof. Gregory means by "narrow vertical range."

(*c*) "The gold is spread through layers which are conformable to the sediments, and is not deposited in verticals or fault planes across the bedding of the rocks, except in the case of a few unimportant secondary quartz veins."

The fact that the gold has not been deposited in "verticals" or fault planes is hardly a proof that the gold is of sedimentary origin. A large portion of the cross faults or "throws" have occurred since the deposition of the gold. As for the secondary quartz veins, gold is of extremely rare occurrence in them.

(*d*) "The gold is distributed in patches and not in shoots."

The facts do not bear out this contention. There are numerous mines on the Rand in which the gold contents of the reef do not vary a couple of pennyweights in many feet, always provided large samples, and not hand specimens, are taken for assay. "Patchy" can scarcely be reckoned a characteristic of the banket reefs. Taken as a whole, they have been noted for the uniformity of their gold contents, and this is one of the principal factors which has led to the extensive exploitation of this system of reefs. Patchy could scarcely be applied to such mines as the Crown Reef and the Geldenhuis mines. In a gold quartz mine, hand specimens taken from the same place frequently show differences in value, even in a well defined shoot of ore, but the vein is not considered patchy for that reason.

The existence of shoots, "chimneys," or vertical bodies of ore richer than the adjacent reef matter and which extend to a very considerable depth, I think, has been fully established on the Rand. First and most important is that remarkable shoot which outcropped in the ground of the Robinson, Pioneer, and The Mint companies. This shoot extended downwards into the claims of the Robinson Central Deep, into the Bonanza (now nearly worked out) and into the Crown Deep: and in all probability will extend into the claims on the dip of these properties. There is another well pronounced shoot which outcropped in the Primrose and its allied properties, and extended into the Rose Deep. The Nigel Gold Mining Company's property is full of shoots, and many instances of shoots occurring in other mines could be mentioned. If it is desired to prove that the gold of the Rand was originally placer gold, some other arguments will have to be brought forward instead af those that the gold is patchy, and that no "chutes" exist in the banket reefs.

(*e*) "The gold tends to occur on the footwall side of the conglomerate beds, or to rest on layers of quartzites, which acted as false bottoms in the reef series. Owing to the redistribution of the gold during its solution, this rule is not so general as in the case of a recent placer; but it appears to be the general experience through the Rand."

As regards the occurrence of the gold on the footwall, I am willing to admit that in many of the banket veins the richest ore is found on the footwall, but there are so many exceptions that it certainly cannot be regarded as the rule. Among the exceptions can be mentioned the Main Reef Leader, which, in the Central Rand while overlying the Main Reef, is so close to it that it forms practically a part of that banket bed, and, notwithstanding this fact, is much richer than the Main Reef itself.

The theory that the gold was deposited with the pyrite, which was precipitated from the sulphur spring solutions, easily accounts for the better quality of the ore on the footwall, as in many instances the lower portion of the banket beds were less silted up, and therefore offered easier access to these solutions than the upper portions of these beds. This is quite conceivable when it is remembered that in sedimentary beds like those of the Witwatersrand, the pebbles would settle before the finer gravel, and would naturally have more open spaces between them, offering channels for the gold bearing solutions.

Prof. Gregory speaks of the redistribution of the gold during its solution. The solvents for gold are not very numerous, and it is not easy to conceive that the metallic gold of the placers should have been dissolved by any solution which may have emanated from the hot sulphur springs or solfataras consequent upon the eruption of large quantities of the basic igneous rock which uplifted the Witwatersrand beds. On the other hand, it is quite possible that these same solfataras brought in sulphide of gold in the sulphide of iron, which was deposited in the banket beds. The sulphide of gold being a very unstable compound would soon, in the presence of air and water, be resolved into metallic gold and sulphuric acid. It is highly improbable that the sulphur springs themselves had the power to dissolve gold. If the theory of infiltration is accepted, as accounting for the presence of gold in quartz veins, there seems to be no valid reason why the presence of gold in the banket should not be accounted for in a similar manner.

(*f*) "The rich patches occur at varying horizons, dependent upon the frequent local variations in the currents that necessarily occur during the deposition of a series of deposits upon a shore."

This has already been answered under the heading II. (*d*).

III. "The objections to the infiltration theory include:

(*a*) The absence of ore shoots."

As ore shoots certainly do exist [see II. (*d*)], this cannot be taken as evidence against the infiltration theory.

(*b*) "The non-existence of the 'verticals' up which the gold may have been introduced."

"Verticals" are certainly not characteristic of all gold deposits, the genesis of which is attributed to infiltration. Moreover "verticals" have something of the nature of shoots.

(*c*) "The limitation of the gold to special seams of conglomerate and its absence from beds of sand and bastard reef which lie immediately below rich banket, and must have been equally open to percolating solutions."

The gold is unquestionably limited to certain seams of conglomerate. It is also limited to certain quartz lodes in formations where the deposition of the gold is unquestionably due to infiltration with iron pyrites and other minerals. Investigations of gold quartz veins have not yet determined why the gold bearing solutions should take any particular course, and if there has been no general rule discovered governing the distribution of the ore in quartz lodes, why should this argument be used to prove that the gold in the banket was originally placer gold!

"Beds of sand and bastard reef," which last I take to mean a coarse-grained sandstone, or as it now is a quartzite grit, did not admit of the percolation of the gold bearing solution. It is certain that compact sand will not permit of the passage of solutions as well as beds of pebbles, and this is most likely the reason that these beds carry very little gold. At any rate, these beds did not offer as free a field for infiltration as the banket beds.

IV. "The essential difference in the distribution of the gold between the placer and infiltration theories is that, according to the former, the gold should originally have been deposited at the same time as the deposition of the conglomerates; whereas, according to the infiltration theory, the gold should have been introduced after the formation of the whole sedimentary series. That the gold was contemporary with the conglomerates is shown by:—

(*a*) "The beds of ore being always parallel to the bedding planes of the rocks."

This has been answered under the headings II. (*c*) and II. (*e*).

(*b*) "The absence of the 'verticals' of South Dakota, and the copper slates of Thuringia."

Refer to III. (*b*).

(*c*) "The presence of gold in the conglomerates before they were cut through by the contemporary erosion, which led to the formation of the 'wash out' channel in the May Consolidated mine."

It does not seem to me that the case occurring in the May Consolidated mine, mentioned by Mr. Ross Browne, proves that the gold was deposited with the banket. There is no reason why the particular body formed from the sand which filled this wash-out should not also be barren. If the gold bearing solutions did not generally impregnate the quartzite, there should be no reason why this particular stratum should be impregnated.

V. The microscopic evidence shows:

(*a*) "There is no evidence of infiltration or the presence of the secondary minerals typical of the infiltration processes."

(*b*) "There has been no conversion of the banket into a continuous sheet of vein quartz."

It cannot be denied that there is not a large amount of vein quartz in the reefs. Nevertheless crystals of quartz in small cavities where there was an opportunity for them to form, are found in the banket, and there is a certain amount of binding silica in the matrix. The varieties of lodes formed by infiltration are so numerous, ranging from the gold bearing vein of almost pure quartz to the silver or copper lode containing the greatest assortment of minerals, that it is not a convincing statement to say that there are any particular minerals or combination of minerals, which are characteristic of lodes formed by infiltration or lateral secretion, as Prof. Sandberger called the phenomenon. It must be remembered that the matrix of the banket is basic, and being basic quartz cannot predominate. If it is a fact that a predominance of basic material proves that there has not been infiltration, then only quartz and other acid lodes could have been formed in this way. It cannot be admitted that certain minerals are evidence of infiltration, and others are not. The fact that crystalline minerals are present in a vein can be taken as evidence that these minerals were deposited from solutions. They may have been deposited from infiltrated solutions or they may have been deposited from solutions formed from minerals already present in the vein, and it is not always possible to determine to which cause their presence should be attributed.

(*c*) "The secondary minerals produced, such as the chloritoid, are typical of pressure metamorphism and have been developed alike in rich and barren rocks. Rich and poor banket, bastard reef and quartzite, are shown by the microscopic evidence to have all undergone the same changes, and the richest material sometimes shows less change than barren material."

This is very likely the case, and there may be very little difference between the banket, bastard reef and quartzite in this respect. The principal mineral brought in by infiltration was the iron pyrites, and this is most predominant in the banket. It is highly probable that the pyrite carried the gold, and, therefore, it can be considered the matrix of the gold. It can be considered more the matrix of gold than quartz, for in gold quartz veins where this mineral (pyrites) is present it is almost always associated with the gold.

VI. "The banket differs from the gold ores due to infiltration in other fields, for example, those of South Dakota: and the best general agreement is with that band of modern beach placers, which extends for 50 miles along the western coast of the South Island of New Zealand. The Kanowna lead is quoted as a case of the solution and redeposition of gold in a modern placer."

Being unacquainted with the Kanowna lead, I am unable to controvert this statement, but in view of the few solvents for gold which are known, the evidence of such a phenomenon should be very carefully sifted before it is accepted.

VII. "The absence of conclusive evidence of any considerable impoverishment in depth is an argument in favour of the alluvial origin of the gold, and is favourable to the further extension of the banket in depth."

This statement means that gold mines give out in depth. This is more or less true, but it can hardly be said to have been proved in all cases where the lodes extended for a great distance on the surface.

De Launey mentions (p. 4) the great abundance of rolled pyrites. The greater part of this pyrites is not water worn, and if a sample is broken it will be found made up of crystals radiating from the centre. The well-known "Buck-shot" ore of the New Rietfontein mine is characteristic of this formation, which resembles that found in nodules of brown iron ore. Moreover, it is hard to believe that pyritic water worn pebbles could be found associated with quartz pebbles which had been subjected to a grinding process such as has given them their present shape.

The variation in value in adjacent parts of the same reef, although not very significant, would argue more in favour of the infiltration theory than it would in favour of the placer theory. But as this argument has been used both *pro* and *con*, it seems to be one of those "rules which works both ways."

It certainly would be a difficult matter to prove that veins due to infiltration are always oblique to the stratification, and as for that it is by no means an uncommon occurrence for the gold to jump from one banket reef to a parallel one. As examples, the ore in the New Rietfontein and the Cason mines can be mentioned. There are, however, numerous similar phenomena in other mines.

Sulphuretted hydrogen was, of course, present in the solfataras, and was very likely due to the decomposition of the iron sulphide dissolved in the alkaline waters of those springs. It is not necessary to account for the presence of iron pyrites in the banket by presupposing the presence of magnetic iron (frequently a secondary product itself) which was afterwards converted into iron sulphide. There is plenty of iron pyrites in the earth's crust which was not formed by the conversion of magnetic iron. It is probable that the precipitation of the gold bearing pyrite was caused as much by the dilution and cooling of the hot spring solutions as it was by other reactions. Nevertheless, to determine what the actual reactions were which caused the deposition of the gold bearing pyrite is hardly possible at the present day, as the elements which composed the solutions were very numerous, and the physical conditions which existed at that time cannot be determined with any accuracy.

Numerous minerals besides pyrite have been found in the banket reefs, among which can be mentioned galena, a silver ore (probably silver glance), zinc blende, calcite, apatite, gypsum and some others, which the writer cannot at present recall, but which at various times fell under his observation. These minerals were not found in any quantity, with the exception perhaps of calcite, but they certainly indicate "vein action," and must have been deposited in the reefs from solutions.

I think it is fairly well established that the bigger the pebble the richer the ore, but this can only be applied to the banket of any particular reef, and there is a notable example of this in the Main Reef Leader of the Aurora West mine, where pebbles larger than a man's head are often found. Some reefs are characterised by small pebbles, for instance, the Rietfontein or Du Preez reef, but this reef, though rich, is a small one. The fact that the ore in any given reef is richer where the pebbles are large can be accounted for on the principle that the large pebbles gave freer access to the gold bearing solutions.

The trouble which I find in accepting most of Prof. Gregory's contentions is that he says the gold has been redissolved and again precipitated, and at the same time almost all the arguments he uses are based on the supposition that the gold is placer gold. If the gold has been redissolved it certainly has been transported from its original place of deposit, for it is difficult to believe that it was redeposited in the same spot a second time. It seems evident that if the gold were in solution this solution would be governed by the same laws which would have regulated its distribution had it been brought in by infiltration, and Prof. Gregory argues that the phenomena of its deposition prove it was originally placer gold.

The President: Mr. Curtis has asked me to express his regret at his inability to be present this evening to read the paper himself. Almost at the last moment he was called out of town to inspect some properties. Mr. Curtis is one of the oldest members of this Society, and it is very pleasing indeed to find him coming forward again at our meetings. To the older residents of the Rand he needs no introduction, because he was one of the men who were most prominent in the early history of these fields. We have to thank him for an able paper on a fascinating subject. This subject, the origin of gold in our banket, has tempted Mr. Curtis to re-enter the fields of scientific controversy. It is a subject which has been before us ever since these fields were discovered, and is likely to be before others when our mills have ceased to crush and our dumps are grass grown. It is of more than academic interest because upon it very largely depends the limit of payable gold in depth. It is very satisfactory to us on these fields to know that there is no conclusive evidence to indicate a material impoverishment as we go down in depth. I move a very hearty vote of thanks to Mr. Curtis for his valuable paper, and I feel sure that some of our mining men will come forward and discuss the paper as it ought to be discussed.

Mr. T. Lane Carter: I beg to second the vote of thanks, and would remark that Mr. Curtis has brought up a subject of vital importance, one which I consider should receive more attention on the Rand than it has in the past. Those of you who have taken an interest in the proceedings of the American Institute of Mining Engineers know that about ten years ago the subject of "Ore Deposits" was brought up in a paper which is now a classic on the subject and which will probably be referred to for many years to come. I refer to the excellent paper by Prof. Posepny on "Genesis of Ore Deposits." One remark he makes in that paper we should take to heart, which is, that the way ore deposits can be studied satisfactorily is for the skilled observer who comes across any phenomena out of the ordinary to carefully note them, and let the world know the facts, because in a mining stope, for instance, an interesting feature which is exposed to-day will, on account of blasting, be gone to-morrow, and unless it is carefully noted and given to the world its evidence is lost for ever. The way to get at the truth regarding the deposition of gold on the Rand is for everyone to give all the data they come across which are of any interest on the subject. I hope, therefore, in the discussion, which is sure to arise on this paper, that those who work underground, especially surveyors and samplers, will bring forward any little point of interest which they may have observed.

The President then called upon Mr. Caldecott to read his paper.

Mr. W. A. Caldecott: The notes I propose to read to-night give a brief account of some observations on works practice made during a recent visit to Mexico. With your permission, I will, at a later date, bring forward the second part of this paper in the form of a discussion on the chemical principles involved in the dissolving of silver from its ores. At present Mr. A. McA. Johnston is kindly carrying out for me some experimental work on this subject, which is not yet completed.

SOME FEATURES OF SILVER ORE TREATMENT IN MEXICO.

By W. A. Caldecott, B.A., F.C.S. (Member).

Part I.—Works Practice.

As I have recently had occasion to pay a short visit to Mexico, an account of some salient features of silver ore treatment in that country may be of interest to members. Whilst there, the courtesy of the managers of some of the most progressive mining companies in the Republic enabled me to visit various reduction works, and to receive much information, some of which is incorporated in the following notes.

The remarkable advance during the last few years in the treatment of silver ore in Mexico may be very largely attributed to the replacement of the old and costly patio process by cyanide treatment preceded by very fine crushing, to railway development and to water power installations, which provide electric power cheaply for remote mines, where the cost of fuel would be prohibitive. Power is now being transmitted up to 170 miles, and supplied at 30s. per horse-power per month, or considerably less than the average cost of electric power on the Rand.

The patio process, invented at Pachuca by Bartolome de Medina in 1557, nearly a century before Van Riebeck landed at the Cape, is still in operation at the same mine in that district, but is now mainly of historical interest. The wretched condition of the mules at present employed in working it, in place of the men who used to tread the torta, makes its disappearance less regrettable than if animals had been replaced by the efficient mechanical digging appliances which have been invented for the purpose.* The process of cyanide leaching preceded by a chloridising roast is also now a thing of the past, and much the same applies to hyposulphite lixiviation.

The pioneer work of our past President, Mr. Chas. Butters, is indissolubly associated with the advance in Mexican cyaniding and fine grinding practice, whilst the El Oro Company is famous throughout the mining world for its enterprise in developing metallurgical methods. But apart from these well-known names, the results of the skill and energy of our fellow workers in charge of various ore reduction plants are obvious in every mining district. Whilst these last are without the great advantages of the cumulative experience and resources of the group organisation, this is in some measure offset by the judicious foresight of individual companies in providing means for carrying on extensive experimental work, and also adequate plants for handling the ore under routine working conditions.

The attitude of the Mexican Government under the firm rule of that remarkable man, President Porfirio Diaz, is distinctly favourable towards the investment of foreign capital in mining and other industrial enterprises. It is to be regretted, however, that in spite of the many opportunities which the application of modern methods in Mexico offers, the amount of British capital now invested there is proportionately far less than was the case some eighty years ago in the days of the United Mexican Company.†

Unskilled labour is cheap, costing in all about 1s. 6d. per day, but this advantage is somewhat offset by its irregularity, owing to the occurrence of the numerous church fiestas. The natural hereditary tendency of the Mexican labourer, or peon, to steal all of value he can, is counteracted by a regular system of special clothing for work in the extractor houses, supplemented by watchmen and electric alarms. Shrines and bull-rings to suit the tastes of workmen appear to be almost regular items in a mining company's equipment, and some knowledge of Spanish is a necessity for those engaged in mining work. In the city of Mexico an electrolytic refinery for gold and silver bullion was started in 1906 by a private company, whose enterprise in this respect is significant.

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* *The Mining World*, p. 693, Oct. 26, 1907.

† *The Mining World*, p. 704, Oct. 26, 1907.

The metric system is generally used in dealing with Mexican silver ores. Thus cyanide consumption is expressed in kilograms per metric ton of 1,000 kgm., or 2,204 lb., and silver or gold values are reported in grams per metric ton. Conversion, however, is readily accomplished by taking 1 grm. per metric ton as equal to 14 grains per 2,000 lb. ton, and 1 kgm. per metric ton as equal to 2 lb. per 2,000 lb. ton. Mexican currency is practically equivalent to one half the value of corresponding U.S. coins. Hence the peso, or Mexican dollar, is worth about 25 pence, or 0·5 dwt. gold.* The fluctuating value of silver, however, introduces a disturbing factor in mining operations with which we are happily unfamiliar here; the result of a 10 per cent. depreciation in the silver market may best be realised when the effect is considered of a possible 10 per cent. drop in the percentage recovery of gold from banket ore.

The average Mexican silver ore is usually a hard flinty quartz containing little extraneous base metal mineral.† The bulk of the value exists in the form of silver sulphide, associated at times with a little arsenic or antimony, whilst occasionally silver chloride and native silver occur. A small percentage of gold is found with the silver and constitutes a considerable proportion of the bullion value, since with silver at 2s. 6d. per oz., gold is worth 34 times as much as silver. As a rule the gold is extracted during cyanide treatment a good deal more readily than the silver, so that special attention during treatment is not paid to it. Manganese is occasionally found associated with silver ore, and is reported to exercise a detrimental effect on the extraction, but I am not aware of any accepted explanation for this effect.

The essential features of present day treatment of silver ore in Mexico consist in very fine crushing of the ore with cyanide solution, and treating the bulk of the resultant pulp as a slime by decantation followed by mechanical filtering, and zinc shaving precipitation of the solution. Whilst the foregoing broad statement may be taken as correct, there exists, as is naturally the case with a progressive art, a considerable diversity of opinion among competent operators as to the advisability of many details of practice, and hence the following more detailed descriptions must not be considered as universally applicable.

The fine state of division and uniform distribution of the values in the ore render finer crushing necessary than in common practice on the Rand, but the means adopted for this purpose naturally vary with the existing equipment of mills and with the views of those in charge. Generally speaking, stamps followed by tube-mills represent the most advanced practice, but at a Guanajuato plant stamp-milling alone with slotted screening equivalent to a wire-woven screen of 45 meshes per linear inch, serves to convert the bulk of the ore into slime, though at the cost of the low duty of 2·7 tons per stamp per 24 hours. At a Pachuca plant stamps, followed by Chilian mills crushing through 80 mesh screening, are used and yield much the same resultant pulp.

The most notable and up-to-date plant, however, is the Mexico mill of the El Oro Company, which has forty stamps and six 4 ft. × 20 ft. tube mills to reduce with 16 mesh battery screening 200 tons of hard flinty ore per day to a product, of which 90% passes a 200 mesh screen. Taking each tube-mill as equal to 20 stamps in crushing capacity, this equipment with five tube-mills running, virtually means a stamp duty of about a ton and a-half of ore per 24 hours. This mill, which has only lately started, should throw much light on the debated economic question as to whether under such conditions all-sliming yields more profit than the present usual production of 50% to 75% slimes, with the balance of the pulp as specially hard and tough fine sand requiring treatment by leaching. Should it be possible in the future to secure as rapid dissolving of the sand values as of the slime values, it is probable that the next operation of separation of solution and solid in these two products will be carried out in the same way as, for example, by horizontal mechanical filters. As is realised here, the question of the actual cost of all-sliming is of great importance in view of the simplicity and other obvious attractions of the method.* Other points of interest at the El Oro Company are the satisfactory use of separate iron guides instead of wooden battery guides, and of concrete mortar blocks. Trials are now proceeding in stamp milling without battery screening but with a high slot discharge on the lines of the old German schubersatz.† As at the well-known Homestake Company in the Black Hills district in South Dakota, a foundry is run by the El Oro Company, and also a small blast furnace for periodically working up slag and other by-products, both of which prove of great service. The corrugated cast-iron tube-mill lining block which was described on p. 60 of the August (1907) issue of this *Journal*, is employed and is claimed to give treble the life of plain iron liners.

Amalgamated copper plates are not generally used in Mexican reduction plants, but following the stamps and sometimes the regrinding appliances as well, Wilfley or other concentrators are employed to remove any metallic or granular particles of valuable mineral, which would yield but a poor extraction in the sand plant. The concentrates are usually shipped to a smelter together with a certain amount of hand picked rich ore.

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* The legal value of the peso is equivalent to 0·75 grm. fine gold.

† Cp. Hofman's "Hydro-metallurgy of Silver," p. 286, 1907.

* See Mining and Scientific Press, p. 658, Nov. 23, 1907.

† Richard's "Ore Dressing," vol. i., p. 214, 1903.

The system of the secondary grinding circuit involving hydraulic classification of all final pulp, with which we are so familiar on the Rand, is not in invariable use, and in general, less attention seems to be paid to minimum water ratio in tube milling, and to maximum practicable weight, drop, and speed of stamps than is considered advisable here.

Sand is commonly separated in vats from the pulp leaving the battery by Butters distributors, and long sand treatment is usual, whilst belt transfer and Blaisdell excavators and aerators are likewise employed at some mills. When the fall and water available permit of it, discharge of sand as well as slime residues by hydraulicking into the creek is practised, and this obvious advantage is little restricted by laws upon the fouling of streams.

In slime treatment, a notable feature is the lengthy agitation and aeration by means of ordinary stirring gear, which sometimes occupies in all fifty hours. This is rendered necessary by the slow rate of solution of the silver.* An ingenious system for the combined aeration and agitation of sand as well as slime, on the principle of a vertical Pohle air-lift pipe in the centre of a conical bottomed vat, 40 ft. high, and 15 ft. in diameter, has been lately introduced at Pachuca from Australia.† In one or two old plants, with shallow rectangular sand vats, converted into slime vats, stirring and aeration are more or less imperfectly performed by means of a small pipe attached to an air-hose, which is plunged by hand below the surface of the pulp into different parts of the vat in turn.

When mechanical stirring gear is used, settlement between the numerous decantations and agitations is not carried to completeness on account of the difficulty of starting stirring again, with the result that although as much as 10 tons of solution per ton of dry slimes are decanted off for precipitation, washing is relatively imperfect, and the strong 0·1 % solution frequently used in slime treatment causes a heavy mechanical loss of cyanide in the solution discharged with the residues. To reduce this loss, which may approach 2 lb. of cyanide per ton of dry slimes, as well as to improve the percentage recovery of the silver and gold, mechanical filters to follow and supplement the decantation treatment are coming into general use. The Butters* filter is most generally employed, though the Ridgway,† Burt,‡ and others are now also being exploited. As elsewhere, clogging of filters in time with lime salts reduces their efficiency and necessitates periodical treatment of the cloths in a dilute hydrochloric acid bath.

In addition to the use of strong solution in slime treatment a good deal of lead acetate** is added to the charge of slime-pulp, varying from 4 oz. to 12 oz. per ton of dry slimes. This is many times the amount of, say, ½ oz. per ton used here; and the apparent reason for the practice is discussed later in the second portion of this paper.

In precipitation practice the variations caused by differing conditions from gold ore treatment are likewise very noticeable. The strong solutions employed and the considerable weight of silver to be precipitated cause the employment of somewhat coarse zinc shavings, for which lead-coating is not needed, as the zinc-silver couple soon formed yields a high precipitating efficiency. As a rule, the extractor boxes are placed several feet above the floor so that in cleaning up, the slimes can be gravitated to a common sump, which supplies the suction hose of the filter-press pump. Simple passing through a fine screen of the zinc-box precipitate affords a product containing half its weight or more of bullion, which is briquetted by air pressure, after partial drying with hot air in the press or in a steam heated pan, and is then smelted into a high grade bullion without any acid treatment. As a plant may yield a ton of bullion in a month, a weekly clean up is common practice, and very large (No. 300) graphite crucibles or forged steel pots 1·5 in. thick are used. Furnaces using crude oil as fuel are in common use, and at the El Oro Company a neat quick-acting air-lift is employed to lift crucibles from the furnace for pouring.

At present the total cyanide consumption varies from two to four pounds per ton of ore and the percentage recovery from all sources of the silver contents of the ore from 70 % to 90 %, and of the gold appreciably more. As might be expected with very fine crushing, and heavy power and cyanide consumption, ore treatment costs are relatively high, and vary between 8s. and 12s. per ton of ore, according to local conditions.††

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* See Griffiths and Oldfield, and Proc. I.M.M., p. 448, vol. xii., 1903; also Trans. Am. Inst. of Mining Eng., pp. 279 and 660. Vol. xxxvi., 1905; also Julian and Smart's "Cyaniding Gold and Silver Ores," p. 79, 1907.

† The Mining World, p. 693, Oct. 26, 1907; also, Mining and Sc. Press, p. 689, Nov. 30, 1907; also, this *Journal*, p. 325, April, 1906.

* *The Mining and Scientific Press*, June 22 and 29, 1907.

† See Allen's "West Australian Metallurgical Practice," p. 176, 1906.

‡ *The Mining and Scientific Press*, p. 717, Dec. 7, 1907.

** Patented by J. S. McArthur and C. J. Ellis in 1893.

†† See Transactions American Institute of Mining Engineers, p. 217, May, 1905, vol xxxvi.

Apart from the fineness and efficiency of crushing, and the reduction of solution value discharged with the residues to a minimum, the chief directions in which future improvement may find scope are probably in reduction of time of slime-pulp agitation and aeration, the more perfect dissolving of the values with lower cyanide consumption, and their concentration in a much less bulk of solution, with corresponding reduction of the precipitation plant. These changes if ever found practicable would reduce power, cyanide, and precipitation costs, as well as lessen the size of plants, which at present greatly exceed in proportion those in operation on the Rand.

Finally, I am sure it will please members to learn that in Mexico, as in the gold mining camps of the United States, our *Journal* is carefully studied, the record of frank discussion which it contains being specially appreciated.

The President: I am sure you will join with me in extending a warm welcome to Mr. Caldecott who has just returned from a long holiday. There are a few members who have taken a conspicuous part in the making of this Society, and I am sure that everyone will agree with me that Mr. Caldecott is one of those few. And as evidence that his energy and interest have in no wise abated, I refer you to the paper he has given us this evening, and to our recent transactions, and, if I may say so on your behalf, this paper is, I hope, merely the forerunner of many others in which he will give us the benefit of what he has seen in his travels. So far as I can gather, his holiday has merely been a holiday in name, for he has covered over 30,000 miles and inspected no fewer than 30 plants in the States and elsewhere. I move a vote of thanks to Mr. Caldecott for his paper.

Prof. J. A. Wilkinson: I have much pleasure in seconding this vote of thanks. This question of the metallurgy of silver is to the teacher a tremendous bugbear, because text books, even modern ones, give innumerable processes, many of which, I believe, are obsolete now, and the ardent enquirer after the truth is left floundering in the mire. The process which Mr. Caldecott has detailed is practically the same as that which is used for the extraction of gold on these fields, namely, crushing in stamp or tube mills, leaching with cyanide, decantation and afterwards precipitation with zinc shavings. It is somewhat remarkable that such a process should be suitable for the treatment of silver sulphide ores, and I think the process as worked out here for gold may claim to be the parent of Mexican silver treatment. The author also mentioned the mining law of Mexico, and in this connection I should like to state that some years ago, when reading a paper on the mining laws of different countries, given before the Institution of Mining and Metallurgy, London,* the author said the mining law of Mexico was one of the best in the world, because it was one of the simplest.

NOTES ON THE ESTIMATION OF CAUSTIC LIME.

(Read at August Meeting, 1907.)

By Edw. H. Croghan (Member).

The Secretary read a note from Mr. J. Gray, pointing out "That on page 183 (middle of the page) the following occurs, 'and its determination by any method would only be a rough guide to cyaniders in arriving at the value of the lime supplied to them.' This should read as follows :—'and its . . . . method would not assist cyaniders in arriving at the value of the lime supplied to them.'"

REPLY TO DISCUSSION.

Mr. E. H. Croghan: In opening the reply to the discussion, I wish to express my satisfaction at the interest that has been stimulated by the paper.

Mr. Johnston is of the opinion that the presence of silicates and aluminates in limes is not proved ; still, the presence of the former is an acknowledged fact, and the latter may also possibly occur. In trying to account for the discrepancies of the gravimetric method, the error was so great, especially in the blue lime, that it seemed quite evident that the formation of some aluminate had to be assumed, if not necessarily such a simple combination as the one employed in the calculations.

As regards the sampling of limes, this is a question which requires very careful consideration so as to ensure satisfaction to all parties concerned. I think this is a matter which at some future time should be thoroughly gone into with the view of trying to fix some standard method of procedure. It is quite useless testing lime samples when a person feels that one is, under present conditions, purely at the mercy of any individual sampler—a state of affairs which makes it practically a speculative venture whether a buyer is getting really a proper value for his money, and *vice versa* for the seller.

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* See "The Mining Law of the United States of Mexico," by W. H. Trewartha James, Transactions Institute of Mining and Metallurgy, vol. ix., pp. 16—23, 1900—1901. *Extract*—"to a mining code for the principles of which the author claims an almost ideal perfection—the present mining code of Mexico."

The procedure Mr. Johnston adopts seems to be pretty thorough, but appears to involve a good deal of time and labour, whereas it is quite within reason that some more expeditious and less tedious method might be agreed on. Personally I do not feel in a position at present to offer any new suggestions.

The deterioration of limes from atmospheric exposure is purely a supposition: still, if the manner in which I have observed limes to be stored on some mines be at all general, it seems to me that especially during damp weather deterioration will take place to quite an appreciable extent. I am therefore still of the opinion that the erection of suitable structures as air tight as possible, is well worthy of consideration. The expense would be very small, and the cyanide manager would be in a better position in feeling that he can, on the basis on which a consignment was bought, use the same amounts, and thus keep his solutions up to the standard alkalinity he requires. As regards the correction of from 0·5 to 1% which Mr. Johnston adds, on the filtration process this is certainly necessary in itself, but if it is to allow for the exposure during sampling, I think it should be possibly a bit more, the variations depending on the weather conditions. I simply mention this, but not for the purpose of letting the seller of limes take advantage of and try and insist upon it, as in consequence of the properties of caustic limes he must allow some degree of consideration. I think if a standard method be adopted, such as the one employed in this paper, the final modifications of which I shall subsequently state, then no corrections need be applied.

Mr. Crosse, I regret to note, absolutely condemns the use of magnesia or dolomitic lime. On our present scanty knowledge of any published data on the subject, this surely cannot be accepted. Cases are believed to be known where equally good results have been obtained with blue limes as with the ordinary white limes generally employed here. The possibilities of the former were especially emphasised by me in consequence not only of their cheapness, but also their contents of manganese oxides. There seems to be generally sufficient caustic lime, which will not only effect slimes settlement, but also the requisite degree of alkalinity, and in this latter connection the magnesia is not a useless factor. Under the conditions of the large volumes of solutions which are handled, even its slight solubility deserves some consideration. I feel sure that collective data on blue limes by those who have had personal practical experience with both kinds would most probably afford a paper of very interesting, useful, and perhaps surprising information, and quite contrary to ordinary theories.

Mr. Taylor mentions the average percentages of his limes at different periods of the year. If other confirmatory data were brought forward, it is quite possible that it would be sufficiently remunerative to lime burners to adopt such precautions for the transport of their material, as would enable them practically alway sto keep up the standard quality of their respective limes.

Prof. Stanley and others are of the opinion that the figures obtained for soluble silica may not be quite correct, and probably are really a little higher, thereby obviating the assumption that an aluminate may be formed, but I think the data are quite reliable.

In stating the method employed, which was open for criticism, it was by oversight not mentioned that after the acid treatment the soluble silica is dehydrated at 150° C. to 160° C. This residue was then proceeded with by dissolving out by very dilute acid the calcium chloride, etc., and extracting the soluble silica from the insoluble residue by sodium carbonate solution, as stated. Quartz sand will not be affected at all.

The corrections pointed out by Prof. Stanley should be as he states: the mistakes were inadvertently overlooked.

Dr. Moir's remarks are chiefly concerned with the slight yet important differences which exist between Mr. Hendrick's work and my own. I may state that it is particularly gratifying to observe the very full abstract in our *Journal* and inserted after my paper, of Mr. Hendrick's detailed work. Still, in view of the points Dr. Moir has discussed between the two papers, there are certain ones I should like also to refer to. These might be considered under three headings:—

No. 1.—The strength of the sugar solution.

No. 2.—The indicator adopted.

No. 3.—The distinction between determining only CaO (caustic lime) in limes, and determining total available alkalinity expressed in terms of CaO, due also to effects of magnesia, carbonate of lime, calcium silicates, etc.

I.—The strength of the sugar solution in Comey's "Dictionary of Chemical Solubilities, Inorganic," is given in a table showing the solubility of CaO in sugar water. As an example, I take the figure that 1·031 gm. of CaO are dissolved by 4·850 gm. of sugar in 100 c.c. of water.

Again, Mr. Hendrick states that whilst 5% of sugar is too low a concentration, there is little difference in the results obtained from the stronger solutions.

Taking the highest figure ·8316 gm. as the true quantity of CaO in 1 gm. of sample, the quantity of actual CaO contained in 5 gm. of the sample is 4·158 gm. From Comey's figures, 19·56 gm. of sugar would be required to dissolve the above amount of 4·158 gm. CaO. This seems a curious point in view of the fact that Mr. Hendrick employed 500 c.c. of a 5% sugar solution which would contain 25 gm. of sugar, and should therefore have been quite sufficient to dissolve the CaO.

If Mr. Hendrick is right, then it follows that Comey is wrong. With regard to Comey's figures, it should be noted that they represent the combined result of the solubility of CaO in sugar water added to that in plain water, but Comey unfortunately calculates a ratio of sugar to lime, which is quite misleading. The true ratio of lime to sugar in the sucrate existing in these solutions can only be calculated after one has subtracted the amount of CaO in solution in the water alone. If this be done, it will be found that the ratio is very near the theoretical, viz. :— 1 : 6·1. The solubility of CaO in water alone at 20° C. is some 1·26 gm. per litre. Therefore for 500 c.c. it would be 0·63 gm. If this amount be subtracted from 4·158 gm., we should have 3·528 gm. left for the 25 gm. of sugar to dissolve, a ratio of just about 1 : 7. As the theoretical ratio is as 1 : 6·1, it would therefore seem that Mr. Hendrick's 5% experiment is wrong, and possibly the explanation might be that not sufficient time of contact was allowed, or the sample was too coarse. On the basis of the above argument, that 1·26 gm. CaO at 20° C. are dissolved per litre by water alone, then, since in my experiments 2 gm. of a sample were used, (assuming it to be 100% purity in caustic CaO), and 1 litre of a 2% sugar solution was used, it is quite possible that a litre of only 0·4% sugar solution would dissolve all the lime. This can naturally be verified by further experiment.

II.—*The Indicator Employed.*—In Mr. Hendrick's method the indicator used is methyl orange on the filtered solution, whereas in my process phenolphthalein is adopted on the original unfiltered solution. With the method stated in my paper, methyl orange cannot be employed, as it indicates total alkalinity, inclusive of carbonate of lime, etc. But phenolphthalein can be safely used. The reason of this is, that as long as we have present actual *caustic* alkalinity, the colour remains, but immediately this is neutralised, the colour disappears, and is therefore the true endpoint. Any reappearance of colour which slowly reappears should not be noted.

This leads to the third point, that is the distinction between true caustic CaO and total available alkalinity expressed in terms of CaO. Dr. Moir suggests the above term "total available alkalinity," and this seems to me the most correct phrase, it being always understood here in reference to CaO, and thus the alkalinity due to the slight solubilities of carbonate of lime, magnesia, etc., is covered by the method. The term caustic lime is therefore not a correct expression for the Rand, although right from the agricultural point of view.

I agree that one can minimise the filtration error by adopting rather the decantation process, that is, after shaking, and then allowing the suspended matters to settle and working on an aliquot portion of the clear solution. The addition of alcohol would be possibly unnecessary if the sugar is first dissolved in, say, half the quantity of water, and then the lime sample added, giving at once a gentle rotatory motion, which should prevent any cake being formed. The alcohol is always a possible source of error, unless it has been previously tested and neutralised.

The prussic acid process Dr. Moir suggests would possibly prove of great service if at some future time experiments were conducted with a view to proving the sugar process. At present I have not been in a position to try it, but hope to at some future occasion, when the results will be placed before you, unless some one will have had the time and facilities to proceed with it sooner.

Mr. Gray's contribution somewhat took me by surprise, and I have perforce to divide it into two sections, firstly, the deprecating and caustic vein, and secondly, that appertaining to the paper proper, which only need be considered. Some points he has raised have previously been referred to. The existence of the literature Mr. Gray refers to does not entitle him to adopt the attitude he took up. The investigations I undertook were not originally intended for publication as they did not seem to me of any great value, but that the paper has borne some fruit is self-evident from the discussions that have taken place, especially in view of Mr. Hendrick's work, which, with one or two exceptions, arrives at the same conclusions as mine, and I therefore do not feel that I have lost any of my self-respect, in that Mr. Gray tries to point out my supposed ignorance as to how investigations of this description should be carried out. The aspersion Mr. Gray tries to throw on the results obtained by me with the sugar method is rather narrow-minded. Perhaps Mr. Gray, who uses the sugar method at present, will vouchsafe us the information how he has avoided, at the time of writing, these same errors himself. It seems a curious point that Mr. Gray does not note the close agreement of the acid and the sugar method. Is this an oversight in not discussing it, or is the acid method correct; if so, is the sugar method I propose a proper one? The obsolete gravimetric method has until recently been employed by his confrères, and it therefore seems to me a peculiar proceeding that the sugar method is now adopted by him. Is it possibly due to the fact that it simply entails very little time and is a very ready and convenient process to adopt, without having definitely proved it first, and is therefore good enough for their purposes. I still remember the time Mr. G. W. Williams suggested the sugar method—when by many chemists it was ridiculed not publicly but privately.

In regard to the theory I proposed, which was the final extract from a long series of calculations, it would have been more to the point, if instead of decrying it as Mr. Gray did, he had rather recalculated a fresh one. Extracts from literature are a simple matter, still it would have done Mr. Gray no harm to do a little mathematical work on the more recent investigations of silicates, etc. The only points of interest and any value of Mr. Gray's remarks are the experiments he carried out. The filtration method certainly seems out of it, and the point seems to rest between the shaking and decantation processes. More experiments are needed, and possibly on totally different lines. At present the method I would propose is as follows:—The lime is all ground to pass through a No. 60 sieve. Transfer 2 gm. of the lime sample through a glass funnel into a litre flask containing 20 gm. sugar dissolved in about 300 c.c. distilled water, giving a rotatory motion as the sample falls into the solution. This prevents any caking taking place, and obviates the addition of any alcohol. Make up to 1 litre and shake for half an hour. Let stand about 12 hours or overnight. Shake the mixture thoroughly for about half a minute or so, and immediately measure out 250 c.c. into a large beaker (600 c.c. capacity). Add 5 c.c. of a 1% neutral alcoholic solution of phenolphthalein and titrate with $N/10$ acid, constantly stirring and taking the end point on the first disappearance of the colour, and not that which often slowly reappears. Each c.c. of $N/10$ acid on the 250 c.c. taken is equal to 0·56% of alkali, calculated as CaO, or 0·34% when calculated as Dr. Moir's pet hydroxidion, a name, which to cyanide men and persons concerned might seem in sound akin to the somewhat terrifying illustrations and names of prehistoric times.

To avoid calculations, take 2·24 gm. of the lime sample, and adopt, say, a 2% sugar solution, then proceeding as above, each c.c. of $N/10$ acid is equal to 0·5%.

In conclusion, I would beg to thank not only those who have discussed the paper on its merits and in an impartial spirit, but also Mr. H. D. Bell for the analytical help accorded to me by him.

The President: I have already thanked Mr. Croghan for his paper, and I think we are further indebted to him now for the discussion which it has drawn, and for his reply thereto.

NOTES ON WAIHI ORE TREATMENT.

(*Read at July Meeting, 1907.*)

By Ralph Stokes (Member).

REPLY TO DISCUSSION.

Mr. R. Stokes: In reply to the comments on my paper I must express gratitude to those contributing to the discussion for having so leniently passed over the many shortcomings, essential to a report based upon the observations of an outside observer. The remarks of Mr. Dowling and Mr. Deakin were, for the most, elaborative of my description, but Mr. Alexander has fault to find with the inadequacy of detail provided, relative to tube mill practice, for sound comparative purposes. He says that he "cannot agree with the author that there is any parallel between his description of tube milling at the Waihi and common practice on the Rand . . . ." I cannot admit that I endeavoured to establish any such similarity between the two fields, for I primarily drew attention to the peculiar hardness of the Waihi's chalcedonic quartz. With a fundamental distinction between the ores treated, it seemed to be unprofitable to attempt any detailed comparison of efficiencies or to provide data which might appear to suggest a superiority of skill on either side. Regarding the "home-made" honeycomb liners, patented by Mr. Barry, it may be mentioned that these had only been in operation for a few months at the time of my visit, a year ago, but I hope to obtain from New Zealand some notes upon their life and general economy, in the light of recent performances, for publication in this *Journal*. With regard to vacuum filters, to which Mr. Dowling and Mr. Alexander referred appreciatively, I may add that the Waihi Company has now largely increased its complement of these machines, after an experience of their functions which should have proved conclusive.

THE LABORATORY: ITS ECONOMIC VALUE.

(*Read at October Meeting, 1907.*)

By A. McArthur Johnston, M.A., F.C.S. (Member).

DISCUSSION.

Mr. A. Richardson: As the author remarks, the quality of the explosives in use here is so excellent that accidents cannot be attributed to faulty manufacture; it is doubtful, however, if we derive full advantage from the energy available.

It is popularly supposed that in the detonation of a charge the force developed in the first cartridge is the same as that developed in the last, but experiments have proved that there is a diminution in the strength of the detonation as it proceeds down the line of explosive, pointing to the necessity of employing a detonator, which by a powerful initial blow will insure the superior detonation of the charge right to the bottom of the hole, an important consideration if the drilling of 10 and 12 ft. machine holes should become the general practice.

In some tests conducted in America it was found that the relative strength of a 60% dynamite was 59·7, 62·2, and 63·3 when detonated with No. 3 (American) (9 gr.* fulminate), No. 4 (American) (12 gr.), and No. 5 (American) (15 gr.) caps respectively, but on the Rand we have no information to guide us as to whether we may look for an increased effect with gelignite and blasting gelatine if instead of the usual English No. 6 detonator (15·4 gr. fulminate) we use No. 7 (23·1 gr.) or No. 8 (30·9 gr.). Those who have had the opportunity of using No. 8 detonators have told me that they never got an unbottomed hole with them, a statement that did not apply when using No. 6. It is important, therefore, that we should have some tests on this head, as the use of an unsuitable detonator leads to an inferior order of detonation, with its accompanying generation of noxious gases, inefficiency and danger.

There is another point worth investigating, that is, the rate of deterioration in the strength of detonators when exposed to mine air, for although detonators are used up rapidly, a box may remain below a good many shifts before it is finally disposed of, with a resulting loss, in a damp situation, of a large percentage in strength, and, as a consequence, poor detonation and its results. It has been found, for instance, that detonators kept in a bottle on damp sawdust failed, after 40 hours, to explode dynamite, a more sensitive compound than either gelignite or blasting gelatine, although they still exploded themselves.

In conclusion, I would like to ask for a reproduction of the Qninan curve in the *Journal*, its equation, and an illustration of the practical uses to which it can be put.

NOTES ON SMALL STOPE DRILLS.

(*Read at October Meeting, 1907.*)

By E. M. Weston (Associate).

DISCUSSION.

Mr. T. Lane Carter: A great deal has been said on this paper, and I have no desire to cover ground already gone over. Mr. Weston, in his paper, gives his experience of small drills up to date, if I can put it that way, and his experience has apparently not been over satisfactory. He mentions the Holman drill, and I can bear out what he says with respect to it. The small drills we have had on the Rand have not hitherto been a success. I have tried these small drills, tried to get a narrow stoping width with them, and after several months found it paid me to take the machines out of the stopes and put in ordinary $3\frac{1}{4}$ and $2\frac{3}{4}$ in. Ingersoll-Sergeant drills. I remember one instance before the war where an efficient miner was put on contract on the basis of the width of his stope and with an ordinary $3\frac{1}{4}$ in. machine he managed to carry the stope 36 in. for a number of months. Of course, conditions were favourable, but it shows that with large machines and good workmen narrow stoping widths are often possible. There is one point to which I do not think sufficient attention has been given. The vital question in regard to small drills is "can they carry a stope narrow enough?" That is what we are aiming for and what we need. It is often the custom of the Rand to consider only from the point of view of the central section, the rich section, but those of you who know the East and West Rand are aware that there is a great difference in the conditions, and that machines suitable for the Central Rand are not so satisfactory in the poorer districts. The question of narrow stoping is one, which is of most vital importance to the poorer sections of the Rand. On the central section there is such a splendid margin of payability that a few inches in stoping width more or less does not mean as much as on the western section, where an average difference of 6 in. in stoping widths may mean the difference between a payable and an unpayable mine, and the narrow stopes therefore become imperative. Personally, I am rather sceptical as to narrow stoping with small drills over an extended period. I know from experience that with a good man working on the bonus system for narrow stoping widths, it is possible to carry stopes in favourable ground, 24 in. and under, but I very much doubt whether any small machine yet invented will carry a stope of this width under similar conditions. I sincerely hope one will be forthcoming, but I have serious doubts. In connection with Mr. Weston's paper I suppose you have read a paper which came out last year in the American Institute of Mining Engineers, dealing with small drills, from which it is interesting to see that the results obtained over there are favourable to their use. There is also the question of air mains and leakages. I think we have at last awakened to the fact that in many mines on the Rand sufficient attention has not been given to this very important point of air mains. The air pipes were put down many years ago, and to put in pipes of the proper size to-day would in many instances involve a large capital expenditure, but where that capital is in hand I think in many cases it would pay to take up the pipes and put down larger ones, as the loss due to inefficient work and extra coal consumption is great. On some mines the direct loss due to air pipes of too small section is from £100 to £200 per month in extra coal consumed for air compression. I am sure the discussion of this paper and of other papers will establish the fact of the necessity of larger pipes and sufficient air pressure in our mines, and in future in our deep level mines more attention will be given to the installation of the air pipes. Mr. Weston mentions electric drills. I believe if an electric drill could be invented which would be efficient it would be a great blessing, but I do not believe there is much scope for a drill which is supplied by air compressed in the mine by an electric machine near by. The only places where such machines would be successful are in countries like the Western States or Italy, where there is enormous water power, and where electricity is very cheap, but until the Victoria Falls electricity gets here I am afraid power will not be cheap enough on the Rand for such machines.

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* The English equivalents are: No. 3, 8·3 grains; No. 4, 10 grains; No. 5, 12·3 grains.

Mr. H. Fraser Roche: In the main I agree with Mr. Weston's ideas of stope drills, and in making a few remarks on the paper, would like to try and emphasise a few of the most important points in it.

High pressure at the machine is a point which most of us recognise the importance of, but are not always able to obtain or maintain, as the requirements of many mills are greater than most compressors can support. The increase in efficiency in drilling, due to higher pressures, is more than directly proportional to the increase in pressure. As Mr. Weston says, it is evidently impossible to make any big installation of stope drills without adequately increasing the compressor plant. While touching on air pressures, I would mention that the management of the Village Deep have during the past nine months carried out exhaustive trials on a new type of machine, the Konomax. Some people claim that they have had this type of machine on the shelves of their curiosity shops for years, and if this is the case, I consider it is very poor testimony to their intelligence that they did not make more use of what is evidently a big step forward in rock drilling machines.

The highest compliment that can be paid to the Konomax, is the fact that the extended trials of the machine have been so satisfactory, that the management of the Village Deep have installed 50 of these machines in their mines.

The net result of these trials shows that, under similar working conditions, they do their day's work as rapidly and as well as any machines at present in use on these fields, and with an air consumption of up to 40% less than other machines. Certainly there are a few small errors in design and manufacture: however, I do not doubt that the Konomax of the future will be even better than that of to-day. Anyway, as it is the most efficient drilling machine at present in use on these fields, it is a considerable feather in the cap of the local inventor. The only other machine of this type is the Minimax, and, so far, this machine has not proved equal to the Konomax: it has several defects in design, the chief being great length of exhaust port. I trust the Rand Drill Co. will try again, and succeed in making a better machine.

Going on to small stope drills, I am not personally acquainted with many of them, and think that these fields have yet to discover a really serviceable one.

I attach very little importance to the stope drill contests, now in progress (it may certainly prove that some drills are quite unsuitable for the work they are expected to perform) and maintain that those that are shown up well have their real tests still to undergo, in trials of six months' duration under normal mining conditions to prove their general reliability, durability, and low cost of maintenance. The Gordon seems to me to be the best I have seen so far, and is now certainly a model of manufacturers' skill. I do not, however, favour a hammer drill for the hard quartzites of our deep levels.

As Mr. Weston says, the Little Konomax is a most ingenious machine, and I was most taken with it: when complete it should prove to be right in the front rank of the stope drills. It has the additional advantage of having an automatic feed. Before leaving the drills, I would like to impress on manufacturers and others the necessity of indicating their machines. The indicator diagram of a rock drill shows up very easily and quickly any defects in the valve motion and internal working of the machine. The cards off the Konomax are now as perfect as those of a big steam engine, and by this means most of the original defects have been removed. Several other types of machines, largely in use on these fields, show considerable wire drawing and compression.

As the author states, we know but little of rock cutting and cutting edges. We have been experimenting a good deal in this line of late, and shall at a later date give some figures of the results obtained. So far, the weakest point in the drilling machine is the shank, about mid-way between point of power and the cutting edge. The drill is out of proportion, and consequently we get much vibration and loss of gauge in our cutting bits; with greater rigidity we can start at 2 in. and finish at 1⅝ in. To effect this, we are having our shanks made up to 1⅜ in., instead of 1⅛ in., and find increases in speed of drilling up to 25%. The employment of big shanks I consider to be one of the most important factors in increasing the efficiency of our rock drilling, and I am certain that this will be shortly taken up universally on these fields.

Another point in conclusion, our cutting edges are too sharp: a greater angle gives us better results. I have seen and tried the Anderson bit, and consider it a most promising tool; it should certainly replace our present system of steels. All the advantages claimed for it by Mr. Anderson are borne out in practice. The increase in drilling speed is, I think, mostly due to the big shanks.

The author, by his excellent paper, deserves our great thanks, and I hope the interest aroused by it may be fully maintained.

Mr. M. H. Coombe: I see that this is the last time this paper is up for discussion, and I would like to take the opinion of members present as to whether the discussion cannot be continued at the next meeting in view of the great interest aroused, and also in view of the recent stope drill contests at the Robinson. A great deal of argumentative matter has accumulated round these contests, and I think it would be better to postpone the final discussion until our next meeting. I have been personally making some notes, as this matter of rock drilling appeals to me very strongly, and I intended to bring them up this evening, until I saw our President's evidence given before the Mining Commission, in which I found some matters which I should like to digest a little more before I come to a conclusion. I am somewhat of the same mind as Mr. Carter. I believe they will have to do a good deal of inventing yet, and bring out a good deal better device than even the Gordon drill, before they will be able to compete economically with hand labour in a small stope. My contention is that if air can be led to a small stope drill it can be led also to a larger drill. My experience has always been in favour of a large drill, and it has shown me over and over again that the big machines beat the little ones. I remember that we were once short of hand labour and I took small machines and two of the best men I could find. They brought the stope from 30 in. with hand labour to nearly 4 ft. I had no false hanging wall and nothing to prevent keeping the stope small. I put in two large machines and did much better work. It is my conviction that where I can work a small machine I can work a large one to better advantage, and if I want to carry a stope economically, I can work anything under 5 ft. with hand labour far more economically than I can work machines, and so can any other man on these fields.

The President: We have already decided to continue the discussion of this paper, and I am very pleased you are going to come forward and give us your views at the next meeting. I promised when Mr. Weston read his paper to say a few words upon it, but it so happens that what I had intended to say in the course of discussion on this paper I said in my report to "South African Mines" and also before the Industry Commission; but I have been asked by several members of the Council to hand that statement in as a contribution to the discussion, and also to read to you the final paragraphs, so with your permission I will do that.

1. ONE MAN STOPING MACHINES.

The competition was confined by the promoters to machines, which, during the trial could be handled by one man. The idea underlying this was presumably, that if a really satisfactory small drill was obtainable, and could be placed in the hands of unskilled white labourers under skilled supervision, or in the hands of selected natives under white supervision, in each case there being one man to each machine, it would be a momentous step in the direction of solving our labour problem. But one man per machine is not absolutely essential for this. It is quite conceivable that the same end could be obtained with two men per machine, and it must be remembered that our coloured labour is comparatively cheap, and the additional cost of a Kafir assistant to the unskilled white man, or of two Kafirs on a machine, one the runner, and the other the assistant, might be more than counterbalanced by exceptionally good work on the part of the drill. Even in the case of the machines which can be quite comfortably handled by one man, the comparative cheapness of our coloured labour may determine that, for the best economic results, they should be run by two men rather than by one. This question as to whether there should be one man or two men to each machine is really of minor importance, but is of interest at this juncture, because a machine weighing $129\frac{1}{2}$ lb. entered the contest, and most of them scaled 100 lb., or over. I believe I am right in saying that in at least some of these cases, the competitors made no pretence of considering their machines one man drills, except for the purpose of this particular competition That weight is a most important factor in the selection of a machine for stopes of 3 ft. and less, requires no discussion ; in such cramped working places the manipulation of the lightest machine is fatiguing to a degree, and great weight is prohibitive, but it appears to me that between this limit and the 4 ft. 6 in. stopes in which our $2\frac{3}{4}$ in. to $3\frac{1}{4}$ in diameter machines find economical application, experience may prove that good machines of, say, even 130 lb. are capable of doing satisfactory work, notwithstanding the handicap of this weight.

One important point on which the trials departed from every day practice in very small stopes, was in drilling four holes from each rig up. We are looking for a machine which will prove a satisfactory substitute for hand drilling in our small stopes—stopes which we would like in most cases to see 3 ft. and under, and which do not exceed 4 ft. 6 in., this latter figure bringing the stope within the sphere of large machines. Now with a good hanging and foot wall, one hand hole will cover the width of even a 3-ft. stope, and, generally speaking, the benches in such stopes are often one hole benches, each hole being pointed to take full advantage of planes of weakness, etc., and each hole being blasted before a back one is drilled. In this way a high duty is obtained, and there is economy of explosives, about 60% of that needed with machines sufficing, a saving which is worth noting. With the ordinary machine, the holes, because of the weight, are located in an arbitary fashion ; lines of weakness, etc., are ignored, and generally speaking, everything is sacrificed to the obtaining of as much footage as possible from the expensive piece of machinery. Thus back holes are drilled before the front holes are blasted, and much explosive is wasted because of the almost unvarying arrangement of the holes, the lines of weakness not being taken advantage of. The ideal machine for our small stopes, is one with an exterior as similar to a piece of piping as possible—and the "Gordon" in this respect leaves little to be desired—and so light that it can be set up, say, 8 or more times per shift without undue fatigue ; the manipulation should be so easy that the principles of hand holing can be observed in every respect, and thus a very high duty and small expenditure on explosives be attained.

In other words, we want a machine to drill one or two holes per bench, and not four holes per bench ; we lose every time on the back holes, and further, whilst on this subject of four hole benches, it is as well to point out that the maintenance of good four-hole benches becomes increasingly difficult as the stopes decrease in width. All this points therefore to the ideal machine having the minimum weight consistent with suitable strength, and the Flottmann and Hardsoeg may be cited as examples of what can be done in the direction of reducing weight, the former being $52\frac{1}{4}$ lb., and the latter $44\frac{1}{4}$ lb. It is questionable whether the Gordon, though scaling only 72 lb., has not overstepped the ideal weight limit, but it is a balanced machine, and its design is such that it is easy to both lift and manipulate. Of course, in reducing weights, the difficulty has, from the first, been to maintain sufficient strength in the various parts, but the improvements being effected in the designs of these small drills is so marked, that a good machine of the ideal weight of, say, 60 lb., will, sooner or later, be in the market.

2. CONSTRUCTIONAL FEATURES.

(1) *Hammer Type versus Reciprocating Pistons*—Both types were represented in the contest. Though, perhaps, somewhat irrelevant to the subject of small drills, it is of interest to note that large machines of the hammer type have been tested on these fields with unsatisfactory results. Thus the Leyner water drill was tried and abandoned here, the reason of its failure, so far as can be gathered, being (1) its inability to drill deep holes, the jumpers giving trouble at depth, (2) inordinate wear and tear, (3) the miners disliked it. On the other hand, it has been said that had the air pressure been higher, the drill would have given a much better account of itself, it having been designed for high pressure. The Shaw drill, also of the hammer type, which goes to the other extreme as regards size and weight, the latter being about 30 lb., and the piston $1\frac{3}{8}$ in. diameter, with a short stroke, has also been tested in a small way on the Rand, but proved to be of insufficient power to deal with the banket. Hammer machines strike less powerful but more frequent blows than those of the reciprocating piston type, and the less intensity of the blows of the former, permits of lighter machines, columns and arms, etc., and does not call for the same care in rigging up that the heavy impacts

of the other type demand, a point which was quite noticeable in the contest. In the one type we have a great number of light blows, whilst in the other, the blows are comparatively heavy, but of less frequency. Weight is essential in the reciprocators, it is a defect which is inherent to the type, and it is difficult to imagine how a really serviceable light machine, say, of 60 lb. of this class can be designed. That satisfactorily proportioned machines of this weight of the hammer variety are possible is more than likely.

The high speed of the hammer type has, here and elsewhere, resulted in great wear and tear of the working parts, and proportionately heavy maintenance charges. It is claimed, however, so far as the Gordon is concerned, that this drawback can be surmounted by using metal better suited for the work, and that the latest consignments of drills will demonstrate this. There is no gainsaying that much can be done by careful manufacture, but this wear and tear is a factor to be borne in mind, more especially as, all precautions notwithstanding, it is impossible to keep grit from finding its way into the drill sooner or later.

The question of losing holes, owing to the jumpers becoming fixed in them, naturally crops up in discussing the types of machines. On this point the reciprocators have the advantage, inasmuch that the very frequent projection and withdrawal of the jumper tends to prevent the latter from becoming "mudded," and further, in case of fitchering, the compressed air assists the operator in his attempts to release the drill. Not that I wish to emphasise this latter point, for my experience with the small reciprocating type, extending off and on, over a period of 10 years, has impressed upon me two points among others, viz.:—their heavy maintenance charges, and their fitchering propensities and small withdrawing power; the first is being lessened by better design and material, the latter remains much the same, and can only be met by higher air pressure, or a design like that of the Chersen. But, generally speaking, it may be said that of the two types, the reciprocators will lose fewer holes than the hammer type, and are much better suited than the latter for deep holes, say, over 4 ft. Holes over 48 in. are however seldom called for in very small stopes.

1. VALVES. 2. ROTATIVE DEVICES, AND 3. FEEDING ARRANGEMENTS.

(1) *Valves.*—Piston (spool) valves, slide valves, actuated by tappets, and ball valves, together with combinations of these, as for instance in the case of the Little Holman, were all represented in the machines which competed underground. The Kimber was the only valveless machine. The ball valves (examples Flottmann and Chersen) are a new feature on this field, and it would be difficult to imagine anything simpler in either action or design, or better calculated to help in the direction of reducing weight. Further, because of their nature, little inertia and negligible friction, it is reasonable to assume that they do not suffer from that insensitiveness at low pressure which is a characteristic of the piston type, nor again, are they so likely to be interfered with by the presence of grit. It is a valve which would appear worthy of the attention of manufacturers of light weight drills. It is worth noting that the spool valve of the Gordon is made of vulcanised rubber, and is both small and light.

The machines fitted with the tappet actuated slide valves, as, for example, the Little Wonder Tappet, Baby Ingersoll, and Little Kid, doubtless, like their large prototypes, have good running powers at low pressures, in consequence of the positive character of their valve action.

(2) *Rotative Devices.*—In six out of the eight machines, the rotation of the drill was effected automatically by twist bar and ratchet wheel, the exceptions being the Gordon, in which this duty devolves upon the operator, and the Kimber, which has no twist bar, but has a ratched wheel actuated by a pawl, moved by an independent piston. Non-automatic rotation is a not uncommon feature of the smaller machines of the hammer type, and it would appear in such cases that automaticness has been sacrificed because of considerations of weight and simplicity. Whether the non-automatic action of the Gordon, for instance, is a strong or weak feature in the design, is a matter on which there will be difference of opinion. On the one hand, it might be urged that the fact that the operator is compelled, on account of the nature of the rotative arrangements, to give the machine the closest possible attention, is calculated to result in better all round work being done with the machine; but others might maintain in opposition to this view, that any piece of mechanism which requires incessant attention and handling, more especially when such mechanism is in the hands of unskilled workmen, black or white, is more likely to give trouble than the machine which will take care of itself to a greater extent. It is one of those points which a twelve months' trial will best decide. It is true that we have an analogy to this handle turning in the case of the ordinary screw feeding arrangement attached to most rock drills, but it seems to me, that if fitchering is to be prevented, and holes not lost, the rotation on the Gordon is going to prove more irksome than manipulating a feed. But if this non-automatic rotative device will do satisfactory work with such attention as it will receive from

the unskilled operators, that is all that is needed.

(2) *Feeding Arrangements.*—The two representatives of the hammer type of machine which ran underground—the Gordon and the Flottmann—differed from the others, the reciprocating class, in having an air piston feed instead of the screw feed, which is invariably used with the latter. There is much to be said in favour of this air feed, for it is simple, speedy, and automatic. The Flottmann was an especially good example of a wholly automatic drill, both the feed and rotation were automatic, and the only work the operator had to do, was to rest his hand on the cylinder of the machine, in order to detect threatened fitchering.

But in designing the latter machine, the makers, so far as the air feed is concerned, have not properly appreciated the hard knocks and general rough usages the machine will be called upon to withstand, but this part of their drill, at present so obviously weak, is, I understand, to be both shortened and strengthened.

The feed screw of the Kimber machine is very exposed, being affixed to the top of the machine, an arrangement which, whilst keeping the centre line of the cylinder low, does not favourably impress me.

(3) *General Design.*—One of the most striking features of the contest, was the rapidity with which the Gordon was erected and adjusted, and its drills changed. On the column and arm, bolts were entirely absent, their place being taken by steel wedges. Much time was undoubtedly saved by this simple and effective device, and it would be interesting to know whether the idea would give satisfaction with the heavier blows of the reciprocating machines.

The fact that the Gordon is balanced about its long axis, and is therefore not likely to twist in one's arms, that it is cylindrical, and is slipped, nose first, into the clamp, and not lifted on to it, is in its favour, these being points which a machine man would appreciate in a small stope. It is also entirely free from projections, and is thus less likely to be harmed by rough usage and blasting.

There is no reason why the mounting of the Flottmann, for instance, should not be equally simple and convenient.

In order to derive the greatest benefit from these small machines, they will, as already pointed out, have to be erected several times in the course of a shift, and, in view of the fact that making the column secure, even in the case of a small machine, calls for very great exertion on the part of the machine man, I think that makers of small reciprocating machines especially, would do well to consider the advisability of so modifying their columns as to call for less exhausting efforts on the part of the operator.

(4) *Dry and Wet Holes.*—Some years back, when the comparatively highly inclined reef of many of the outcrop mines was being worked, a system of stoping almost exclusively practised was the underhand, a method of work having advantages, but in which the broken reef lodged on the working face, and often caused inconvenience and delay. At the greater depths of the present day, the dip of the reef is steadily approximating to 30°, and our stoping is becoming more and more of the overhand or the breast variety, underhand stopes still predominating here and there, owing to special circumstances or high dips, as, for instance, on the Randfontein mines and the East Rand.

In both overhand and breast stoping, the broken reef is flung by the explosion more or less clear of the working face, with the result that misfires are easily seen and drilling can be resumed with the minimum delay. In all three systems of stoping above mentioned, the holes on the Rand are down holes, and are, of course, drilled wet, thus advantaging the machine, or the boy, as the case may be. In some parts of the world, dry "uppers" are common in connection with overhand stoping, but where such is the case, the conditions are not comparable to those on the Rand.

The above explanation is necessary in view of the fact that the task given the machines underground was the drilling of an equal number of so-called dry and wet holes. The number of dry holes we are at present drilling in our stopes is, expressed as a percentage, so small that it may be ignored, and it is difficult to imagine how this percentage can be increased with advantage. In the trial, the upper holing resulted disastrously for all the machines, with the exception of the Gordon and the Baby Ingersoll, though the performance of the latter was much behind that of the premier drill; but it is only fair to mention that all the other machines would have done better than the figures expressed, had it not been for the behaviour of the drill steel. With the Kimber it was a clear case of the steel being of a section too light for the blow: in other instances bad tempering resulted in the chipping away of the cutting edges and reduction of gauge, making it impossible for the other drills to follow, whilst in other cases the shoulders of the cutting edges were rapidly worn away, resulting also in reduction of gauge and loss of the hole. But, apart from the drill steel, the rate at which these upper holes were drilled, was very slow indeed, compared with the pace on the wet holes, and it was obvious that the drilling of dry holes by these machines would be a failure from an economic standpoint. The Gordon was really the only exception, drilling the uppers apparently without difficulty, and almost as quickly as the down holes. It was the only machine which used water in the uppers, and there is little doubt that this had much to do with the successful accomplishment of its task. It is instructive to place on record that the Gordon drill steel retained its cutting edge and gauge admirably; this, to some extent, is undoubtedly due to the light blows, and the comparatively stationary position of the jumper, but that water also has an incidence on this matter was proved by the fact that the Flottmann steel failed, and that the drills of the other machines, which gave satisfaction in the wet down holes, were very unsatisfactory in the dry up holes, the drills, I was assured, being of precisely the same steel and tempering. In view of this excellent performance claimed for machines with air blast, drilling dry—the Climax Imperial $1\frac{3}{4}$ in. diam. machine, for example—it is a matter for regret that one of this type did not come forward and afford an opportuuity for comparing speed of drilling and behaviour of steel. The health aspect cannot be left out of account in this connection, and the use of water with all drilling should be compulsory. Messrs. Hosken & Co. claim that by using their water spray with the air blast, all dust is allayed. The Gordon, by passing the water to the cutting edge, has adopted the ideal principle of preventing the formation of dust, but, of course, any drill can be provided with a dust allaying attachment, or, on the other hand, a jet of water, quite independent of the drill, may be played about the collar of the hole. So that, from the humanitarian point of view, there is little to choose between the various machines; the dust can be allayed in every case. One point in connection with the water arrangements of the Gordon calls for remark; the spray rebounding from the working face, or coming out of the hole or chuck, was such as would undoubtedly result in the operator's clothes being decidedly damp at the end of a shift of ordinary length.

In leaving this section of the subject—dry holes—which, having little incidence on our operations, is of comparative unimportance to us, it is as well to call attention to the great strain on the machine man which this upper holing entails.

(5) *Drill Steel.*—It is now beginning to be admitted that one of the points in which there has been false economy on the Rand, is drill steel. Quality has too often been a consideration quite secondary to the cost per lb., and further, it is also dawning on us that insufficient attention has so far been paid to the tempering of our drills, both hand and machine. This contest proved an excellent demonstration of the advantage to be derived from sharp drills as against blunt ones, and from drills which retained their gauge as against those which speedily lost their shoulders, resulting in the inevitable trouble with the followers. This was very strikingly illustrated indeed in the case of the dry holing, and, as I have already remarked, many of the machines would have done materially better, had it not been for the defective drills. Several of the competitors retired from the dry holes on this account.

(6) *Effect of High and Low Air Pressure.*—The poor performances of several machines, both large and small, on this field have been largely attributed to insufficient air pressure. It is an open secret that our air pressure underground on the Rand leaves much to be desired, and that what with pipe lines badly laid, and much too small, leakages and inadequate compressor power, many of our machines have been working under most unfavourable conditions.

Now that we are cognisant of this, it is to be hoped that the time is not far distant when every machine will be supplied with reasonable driving power. Some drills—large tappet drills for example—will do a certain amount of work satisfactorily, with air of even 30 and 40 lb., though they would, of course, have a higher duty with a higher pressure; others, however, demand high pressure—60 to 100 lb.—for anything like satisfactory performances. The small drills, as a class, call for higher pressure than large machines, and of the former, the hammer type requires higher pressures than the reciprocating type. Many makers of the hammer type ask for 100 lb., and over, for good results. The trials underground were carried out with air of 60 lb. pressure at the machine, and there was evidence that good work could be done with such pressure by both types of baby machines. The majority of the mines on this field are working below this pressure, and it will be a costly business to raise the standard to it; to carry it above it, say, to 80 or 90 lb., would be very expensive, both as regards capital outlay and running charges. The higher pressure would entail larger compressor installations, and we would experience greater loss in the way of leakage. Many of our properties might advisedly be furnished with large or duplicate pipe lines.

Would it pay to increase the pressure to 60 lb. and more at the machines underground? There are few mine managers nowadays who would think it advisable to deny 60 lb. to their machines, if such a pressure could be supplied, but regarding pressures exceeding this, opinions will doubtless differ. Would an adequate return be given in the form of footage drilled for the increased cost of the air, or would such an innovation merely result in shortening the hours of labour of the machine man? It is difficult to say. Coming back to small machines, there is no doubt whatever that they would give increased footage with the higher pressures, but, on the other hand, it is equally beyond doubt that the wear and tear would be seriously augmented, and that grit would become a more destructive factor. Experience may possibly prove that, viewed from any economic standpoint, 60 lb. to 70 lb., at the machines, is the happy medium; it is a case of trying to strike the mean between the inordinate wear and tear associated with high pressure, and the small footage which will be accomplished if inadequate motive power is furnished.

(7) *Result of Wear and Tear.*—With a new machine of the large type, a miner can often do his four holes in but little over 60% of the time he would need with an average old machine, and not only so, but he goes off shift much less fatigued in the former case. The operator's troubles make their appearance as the machine becomes worn, and it may be taken for granted that what holds good in the case of the large machines will also pertain, in some degree, to the small types. The air losses also increase with the wear. This is a consideration which it would be unwise to ignore in connection with the present competition; all the machines might be described as new, for such as were not actually so, had been made the equivalent. So the competitors had all the advantages attending the use of new machines.

(8) *Incidence of the Man at the Handle.*—It is almost unnecessary to point out that the operator is a very important factor in any calculation of a machine's capabilities. In this competition, naturally, the men were carefully selected by the agents, and we may assume that the latter looked upon them as expert machine men. These men during the trial, worked under competitive, and not normal, conditions: in other words, they begrudged no effort, and spared themselves no strain calculated to help themselves and their machines to the first place in order of merit. The men were of good physique, in some cases exceptionally so, but every man of them was obviously exhausted at the end of the shift, and the four days' run, two days on the surface, and two underground, resulted in a condition of physical strain which must have taken several days to recover from. Of course, such exceptional efforts could not be kept up in every day practice, and the competition results do not therefore truly reflect what these expert machine men would do under normal conditions. How the machines will behave in the hands of the average unskilled whites or Kafirs, it is difficult to forecast. I am of opinion, that fitchering and "mudding" is more likely to happen with these small machines than with our large ones, and therefore, that they call for greater skill, or, at any rate, greater care than the latter, for successful runs. But careful selection of operators, be it white or black, and intelligent and patient teaching, would do much towards surmounting this obstacle; many of our natives are apt pupils when it comes to machine work. But here again it is obvious that only after the machines have been in the hands of hundreds of such operators for twelve months or so, will the true average capacity of the machines, under Rand conditions, be known. The performance of the Gordon machine in the hands of a native since the competition, is a matter on which the Gordon Drills Co. is to be congratulated, for it is promising achievement, even after making all allowances for the fact that the boy was specially selected, and worked under exhibition conditions. A very good performance is claimed for the $1\frac{3}{4}$ in. Climax Imperial Drill, in the hands of a Kafir.

(9) *Conclusion.*—If the down holes only are taken, and these represent the work which the drills will be called upon to deal with in practice, the order of merit is as follows:—

	Average in. per min.
Gordon	1·82
Chersen	1·44
Little Holman ...	1·21
Baby Ingersoll ...	1·16
Little Kid	0·82
Little Wonder (tappet)	0·68
Flottmann	0·56

(The Kimber drilled no wet holes).

In the case of these wet holes, the Little Kid and Flottmann lost time on their drill steel, whilst the Little Wonder dropped behind in consequence of the early exhaustion of its operator. The Gordon, Chersen, Little Holman, and Baby Ingersoll, made very satisfactory runs, and the above figures probably represent approximately their capabilities under the conditions which obtained at the trials.

It is unfortunate that the drill steel interfered so largely with the performances on the back stope, but I have no hesitation in saying that, under the rules of the competition, the Gordon would have maintained its premier position in any case.

This drill is of the hammer pattern, to which, in view of developments, I believe we must look for the ideal 60 lb. stoping drill for our present hand stopes. It is too much to expect that a serviceable reciprocating machine can be made of this weight, and though greater weights can admittedly be handled in even 3 ft. stopes, yet so laborious will their manipulation be in the cramped space, that, other things being equal, the lighter machine cannot fail to win favour in the end. But in this connection, it would be unwise not to recognise the apparently satisfactory work the Little Wonder (118½ lb.), the Little Kid (102½ lb.), and the Little Holman (97½ lb.) for instance, have done in quite small stopes. The all round improvement which has been effected in the design of small machines generally, is most marked, and augurs well for future possibilities.

Regarding the decision of the Advisory Committee to award the trophy exclusively on speed, it was perhaps somewhat difficult to adopt any other course under the circumstances, and certainly speedy drilling is a matter of the first importance in our machines. In fact, so vital is the question of footage that a machine eminently successful in this direction must indeed have grave defects in its design, if it is not accepted as the best machine for our mines. It is not difficult to see that a full appreciation of this point has led to the design of the Chersen and Kimber, wherein power is very liberally provided, in the hope of obtaining a satisfactory return in the shape of very rapid drilling. The Kimber was unfortunate during the trials, but the behaviour of the Chersen must have afforded considerable encouragement to its designers. Both these drills, the Chersen especially, are of great interest, because their power approximates to that of the ordinary 2¾ in. machine, whilst their weight is less than 50% of the latter, and, therefore, in developing these machines, the inventors might perhaps advisedly consider the possibilities in connection with both large and small stopes.

The review of these two machines, with their large piston areas naturally raises the question of air consumption. It is a matter which has, I think, been unduly laboured, for it is largely academic. It is true that it is as well to be cognisant of the air requirements of a drill, just in the same way that it is desirable to know the maintenance charges, but to imagine that primary importance is to be attached to air consumption in selecting a drill for our mines is altogether a mistake, in view of the fact that extra expenditure on air may be more than returned in the form of additional footage; a few extra inches per shift balances the cost of a considerable amount of air. The compressed air is merely one of many charges, and it is immaterial, for all practical purposes, whether it forms a small or large percentage of the total costs. The all important point is that the total inclusive running charges should bear comparison with that of our hand labour. There are, however, two conditions under which the question of air consumption would assume importance, the first being, where in two drills, otherwise equal in every respect, one consumed much less air than the other, in which case the former would naturally be selected, whilst the second is where a mine has insufficient compressor power, and is unable to obtain more, in which case the machine which is called for little air might possibly be installed, regardless of other considerations. But whilst it has become common knowledge during the past few months that many of our compressed air installations are not what they ought to be, it is inconceivable now we are cognisant of the fact, that this state of affairs will obtain much longer.

Only 12 months' experience on a large scale with these small stoping machines will disclose their defects and tell us the full story of their weaknesses and advantages. Such a trial will afford reliable data regarding maintenance charges, the fall of efficiency due to wear and tear, the sufficiency or otherwise of suitable operators, and above all, the inclusive cost of running. With the 200 Gordons, just to hand, running on the Rand Mines, Ltd., and the Little Kid and others already at work, it will not be long before the above important points are known quantities.

In the foregoing remarks it will be observed that I have dwelt but little on the financial aspect of this small machine question. A machine may be very cleverly designed, and leave little to be desired as a piece of mechanism, but unless it can do the work on terms which our mines can afford it is of no value to the industry, except perhaps indirectly in the direction of easing our labour difficulties. In other words, unless these small drills can do the work at an inclusive cost which does not exceed that attending the present hand labour, the mines cannot reasonably be expected to adopt them; but, on the other hand, it is obvious that for any machine which can demonstrate its ability to do the work at the price, there is a large reward in store in the form of extensive sales.

What is the financial aspect of this small drill question at the present day? It is venturesome to forecast events, even under ordinary circumstances; in connection with this country of kaleidoscopic changes it is especially hazardous, but cannot always be avoided. The failures of the small machine in the past may be ascribed largely to the breakages and heavy maintenance charges, poor material, general unsuitability of design, and to the labour charges being on the same costly scale as those of the large machines. Are the machines at present before us going to do any better? Can they do the work with reasonable wear and tear, and thus keep down maintenance charges, and prevent labour standing idle though drawing wage? How are they going to work in the second year as compared with the first twelve months, so far as maintenance and efficiency are concerned? Can runners be found for them, either unskilled whites or natives, in whose hands they will drill an average of at least 8 well placed 3 ft. holes, or the equivalent, per shift? Will unskilled white runners be content to receive 10s. as a shift's pay, or a scale of remuneration which will not handicap the machine's performance to a greater extent than this figure? Can the inclusive cost attending small machines be brought down to a figure not greater than that associated with hand drilling? The advance exhibited in the design of the present machines, and the possibilities it foreshows, the economic revolution which has taken place during the past 6 months in connection with our white labour, inclines one to answer the above questions in a manner favourable to the present machines, or their early successors. The open minded attitude of the mines in this matter is, I think, commendable. After having tried, during the past 10 years or more, machines of many promises, but dreadfully poor performances, there is reasonable ground for scepticism on their part, and for a policy of making haste slowly, more especially as the perfect small drill has yet to be produced. It now lies with the manufacturers to arrange with the mines for, say, a 12 months' trial of their machines under normal conditions, and it is sincerely to be hoped, for the sake of the industry and all concerned, that such a test will result in a really satisfactory machine being discovered.

With your permission I would like to add just a few figures. They are figures which bear upon two of the most important points we have had before us for some years past, dealing with the subject of small drills and bearing upon the labour and cost associated therewith. We have had flying about during the last month or two some rather wild statements regarding the incidence of these machines upon our native labour, and the figures which have been hurled broadcast have varied from 20,000 to 50,000, and I believe in some cases more. But what is the position? Let me say, in the first place, that I think it is possible that with eight 3 ft. holes well placed, these small machines can give us per shift about 5 tons of rock, and seven boys with holes well placed can give us the same tonnage, which works out at 0·7 ton per boy per shift.

You start, therefore, with a basis of seven hand drillers equal to one small machine. Now I may tell you that the number of stoping boys at work on these fields at the present time is about 36,000, so that in the first place it is pretty obvious you cannot send out 50,000, and of these 36,000 you certainly cannot dispense with all of them. You must retain some as stopers; I will tell you why. We have in quite a number of stopes insecure hanging, or where there is merely a thin parting between that particular stope and the stope above or below. Now it is unsafe to run even a small machine in a stope like that, for it might very soon bring the hanging wall down on you. In such a stope you have no option but to use hand drillers, and that is one reason why you must retain some hand stopers. Another reason why you must retain them is for scaling the hanging wall or for taking up leader in the foot. Another is for putting in short holes for breaking up large chunks of rock in the stope or for trimming the stope. You must remember that these small machines will probably do 5 tons of rock per shift under favourable conditions, but you cannot expect them to do their full duty unless you give them good benches, and so you would have to retain a few stoping boys to trim these benches now and then. These are just a few of the many reasons which will call for the retention of a certain number of these 36,000 boys. I have estimated that about 8,000 boys will have to be retained for the purposes enumerated. That leaves us with a total of 28,000 boys, whose services may be dispensed with. If you take that number and divide it by seven, you arrive at the number of machines we can take, viz., 4,000 if these machines only work a single shift. But every machine, of course, works double shift. You divide the number and you get 2,000 machines working two shifts as being equivalent to 28,000 hammer boys. Therefore there is a possible market for 2,000 satisfactory small stoping machines. Let us see how the running of this number of machines would affect our labour. Take the cost of running these machines with white unskilled labour, one man one machine and with one superintendent also for each ten machines. That gives you 4,000 unskilled whites to run the machines and 400 superintendents, total 4,400 whites, and if you take the gangers at present engaged with these 28,000 hand stopers, you get that number reduced to between 3,000 and 3,500. So that by installing these machines and running them with white labour you find work for 3,500 extra white men. Now I believe that no matter how small these machines may be it will be wise to let each white man have a Kafir assistant. If he does not get a Kafir assistant that machine will be standing idle quite a large part of the shift, I can assure you. That would call for 4,000 natives being retained which will reduce the figure to 24,000 natives. It is customary in our hand stopes for the stoping boys themselves to do quite a lot of lashing. It may be that you have 30 hammer boys in one stope, and if you displace these with machines you have to allow for additional lashing boys. Therefore I figure on another 3,000 boys as having to be retained. Therefore, out of the 36,000 stoping boys at present at work you can only dispense with the services of, say, 21,000. Taking now the case where the machines are run by two natives per machine per shift, and one white superintendent per four machines, the 2,000 machines call for 8,000 natives and 1,000 skilled whites. This item of 8,000 natives brings the 28,000 down to 20,000, whose services can be dispensed with. But here again you have to allow for additional lashing, and therefore you can only really dispense with about 17,000 natives. Therefore, if you run these machines by natives, you may possibly dispense with 17,000, if you run them with whites, you may dispense with about 21,000, and, speaking generally, the machines have an incidence on our labour supply of about 20,000. The white labour complement in the case of the Kafir run machines is practically the same as with hand stoping. What about the cost? Will it work out at the same price as our present hand stoping? I will just give you a few figures, though I wish to emphasise the fact that figures are very supple, and I do not attach much importance to them, and you must take them for what they are worth. I have taken, for instance, two boys per machine per shift and one white man superintending four machines. We have assumed that each machine will average 5 tons per shift in the hands of the 4,000 unskilled whites or in the hands of 8,000 natives. I do not say it is impossible they will average that. It is not necessary in comparing the cost of machine work and hand labour to compare every item, you need only take certain items, and that I have done. Where you have two boys per machine per shift and one white man to four machines, I have taken the maintenance and redemption, drill sharpening, etc., and divided them by the tonnage, and I get 4s. 6d. per ton.

If I run the machines with unskilled whites, allowing 10s. per shift for the white man, and giving him a Kafir assistant and allowing for additional lashing, I get the figure of 5s. 5d. per ton as compared with 4s. 6d with natives. Turning now to hand labour with our present force I get the figure 5s. 4d. per ton. By running the drills with unskilled whites you thus get 5s. 5d., a difference of 1d. against the machine. In conclusion, I may say that I base these figures upon ·7 ton per boy per shift, that being calculated upon a 3-ft. hole; but certain of our boys are increasing in efficiency, and you may get a higher tonnage, and if it so happens the tonnage is materially increased, then it goes against the machine. If, on the other hand, the labour which we obtain for running these machines—assuming that we get the machines at all, and it is by no means certain — turns out better work than we are anticipating, then the cost is in their favour. I may say I think our boys are too well paid, and if we grant for a moment that the wage per boy is reduced, there again you get an interfering factor, because the hand stoping would be done at less cost. Turning to the wage per shift for white labour, I do not say that you will not get men here to accept 10s. a day; you may even get the number required in a comparatively short time, but it is one thing to persuade them to accept 10s. a day, and it is quite another matter for them to be contented with it, and it remains to be seen whether they would be permanently satisfied with this rate of pay. I am afraid they would be a thorn in the side of the mine manager.

The meeting then closed.

Contributions and Correspondence.

THE FALSIFICATION OF NITRO-COTTON WITH MERCURIC CHLORIDE AND ITS DETECTION.

Mercuric chloride is frequently added to wet collodion cotton, ostensibly to prevent its becoming mouldy, but sometimes also to make it pass the official heat test, even when it is really defective.*

Ordinary tests for mercury fail because $HgCl_2$ cannot be extracted (except by volatilisation) owing to its affinity for organic matter. I have therefore invented the two following simple processes:—

(a) *Combination of Hargreaves and Rowes' Method with E. A. Mann's Method.*—The moist cotton, along with a piece of (ignited) silver foil is placed in a flask immersed in boiling water, and air is aspirated through it, and then through potash bulbs containing 2% sulphuric acid. After two hours the $HgCl_2$ which has escaped the silver is deposited thereon by using the foil as anode in electrolysing the dilute sulphuric acid from the bulbs (2 hours with about 4 volts required). The foil is dried and sublimed on to a microscope slide, as in the other methods.

(b) *Conversion into non-volatile compound.*—The cotton is extracted with hot very weak KI solution, and the extract (containing K_2HgI_4) concentrated to a small bulk and electrolysed.

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* See this *Journal*, vol. viii., Nov., 1907, p. 156, and Dec., 1907, p. 190.

Iodine separates on the platinum basin used as cathode, and the whole of the mercury collects on the gold or silver foil anode after two or three hours, and is worked up as above. The latter method is the quickest and most sensitive I have tried as yet, and has the advantage of having no danger of explosion attached to it.

James Moir.

January, 1908.

Queries and Replies.

No. 1.—"Can any member recommed any particular form of agitator to handle slimed pyritic (85) concentrate? There will only be 50 tons per month to handle. Prime cost is a consideration, and we wish to avoid the use of compressed air, although this is not a rigid condition." H. C.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Qualitative Analysis of Nickel in Cobalt.*—"We are aware of the very great difficulty of detecting nickel in presence of very large masses of cobalt salts, and we may say that this is one of the most delicate problems of qualitative mineral chemistry. Hitherto there was no specific and sensitive reaction for nickel. The proposed method is based upon the following reaction. Nickel molybdate, which can be obtained by double decomposition of a nickel salt and alkaline molybdate, is insoluble in a neutral or slightly acid aqueous solution containing a sufficient excess of alkaline molybdate. Cobalt molybdate, on the contrary, is exceedingly soluble in the same conditions. As for formation of nickel molybdate by double decomposition, it does not seem to have been hitherto indicated. No mention is made of it in Moissan's chemical treatise, nor in the dictionary by Wurtz. The preceding observation is applied as follows:—Sulphides of cobalt and nickel, obtained by the general method of qualitative analysis, and, in particular, the Carnot method, or simply pure nickel and cobalt salts, or pure nickel salt, are dissolved in nitrohydrochloric acid; the aqueous solution is almost exactly neutralised with an alkali, care being taken to preserve a slight mineral acidity; an excess of saturated ammonium molybdate aqueous solution is added; finally, all is heated gently to about 70 C. while agitating. In presence of cobalt the solution acquires a pink tint, and if there is nickel there is immediately, or in a few minutes, a very heavy crystalline greenish white precipitate, more or less abundant. In the absence of nickel the solution remains perfectly limpid. Formation of a precipitate is absolutely characteristic of nickel in the absence of other metals than cobalt: zinc, cadmium, manganese, etc., also precipitate, but as there are excellent means to eliminate them, there is no inconvenience. If requisite to follow up the characterisation of the elements filtration is employed, and all the cobalt is found in the filtrate and the nickel molybdate is left on the filter. It is washed with a saturated solution of ammonium molybdate, and it is easy to identify the nickel after eliminating the molybdenum by some process. This method enables identification, with absolute certainty, of very small quantities of nickel in presence of cobalt. Thus an inexperienced person can detect 1 cgm. of nickel in presence of 500 times its weight of cobalt in a few minutes. None of the hitherto known methods attains this result without great experience and considerable loss of time. If the cobalt is in form of a cobaltamine salt it will be immediately precipitated, even cold, by alkaline molybdate. This, in fact, is the reaction proposed by Mr. Carnot to characterise cobalt. In this case, and only in this case, the reaction would not be applicable: but it is to be noted that cobaltamine salts are readily transformed into cobalt salts. This reaction also gives a microchemical process for detection of nickel which presents a sensitiveness greater than all the other methods proposed. Nickel molybdate is crystalline, and in form of very distinct small square lamellæ, the corners of which are often slightly decomposed. This method is the first to enable microchemical identification of nickel in presence of cobalt, and from the microchemical point of view is one of the most sensitive. It is well, however, to note that zinc molybdate greatly resembles nickel molybdate; cadmium and manganese also precipitate but cannot be confounded with it. To conclude, let us observe that pure cobalt salts, sold in commerce, contain a small percentage of nickel. Thus, if 1 gm. of pure Merck cobalt, dissolved in 1 c.c. of water, is treated with an excess of ammonium molybdate, a very light greenish white precipitate is obtained in less than an hour. Addition of 0·0005 gm. of sulphate of nickel to the filtered solution almost immediately produces an abundant precipitate."—*Comptes Rendus de l'Académie des Sciences.*—*London Mining Journal*, Oct. 19, 1907, p. 472. (A. R.)

Prevention of Rusting.—"According to the electrolytic theory, when a plate of iron is immersed in water the rusting that takes place develops positive and negative areas of greater or less extent relatively. It is possible to conceive two cases as occurring among all the possible arrangements of positive and negative areas. The first case would be that of a positive spot surrounded by an extended negative area, and the second would be of a negative spot surrounded by an extended positive area. At the positive areas iron will pass into solution and be rapidly oxidised to the loose colloidal ferric hydroxide and oxide which are known to migrate, under the influence of electrolytic action, to the negative polar areas. In the progress of the rusting under water it would be expected that in the first case the ferric hydroxide would be piled up in a crater with the metal eaten away at the centre, while in the second case it would be expected that the hydroxide would form in a conical mass with the surrounding metal eaten away. A low power microscope clearly shows that these two formations do occur on every sheet of iron that has rusted under the water.

According to the electrolytic theory, all substances in solution which contain hydrogen ions should stimulate the rusting of iron, and all substances which develop hydroxyl ions in solutions to a certain extent inhibit the rusting, and, if the concentration is sufficient, absolutely prohibit it. This application of the theory is borne out by the behaviour of acids, salts of strong acids and weak bases (which hydrolyse

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[*] See this *Journal*, vol. viii., Dec., 1907, p. 192.

in solution or decompose to an acid reaction), and by the alkalis or salts of strong bases with weak acids.

It has been long known that solutions of chromic acid and potassium bichromate inhibit the rusting of iron. Experiments have shown that iron immersed in a $\frac{1}{20}$ normal or stronger solution of the latter will not rust. It has been observed, too, that if a rod of bright iron or steel is immersed for a few hours in a 5 or 10 per cent. solution of potassium bichromate, and removed, thoroughly washed and wiped and then immersed in water, rusting is inhibited for a length of time averaging from a few hours to weeks. Cushman explains the effect as one of polarisation produced by the separation of oxygen from the solution and its retention on the surface of the metal in some way without the formation of an oxide. If, as the electrolytic theory assumes, rusting is due primarily to the exchange of hydrogen ions, iron in the condition of an electrode, polarised with oxygen, should be protected from electrolytic action for a greater or less time depending on the amount of polarisation. There is little doubt that all conditions which inhibit electrochemical action also inhibit rusting of iron, which fact in itself argues for the electrolytic theory." —Prof. W. H. Walker.—*Iron and Coal Trades Review*, Oct. 25, 1907, p. 1565. (A. McA. J.)

Insoluble Silicious Residue.—"The use of the term insoluble is exceedingly vague, and in order that different chemists may arrive at concordant results, it is necessary to adopt some uniform method for its determination. The author calls attention to three points in the determination: (1) merely putting an ore into solution does not necessarily throw out all of the insoluble matter, and the quantity of insoluble matter throughout will vary greatly according to the time of treatment and strength of acid; (2) that it is necessary to evaporate to dryness to obtain constant results on the same ore: and (3) that all of the insoluble silicious residue is not thrown out by one evaporation and that a second evaporation of the filtrate is necessary to obtain all of the silica and other insoluble matter.

The author's definition is that 'insoluble silicious residue' is that portion of a mineral substance remaining insoluble after the following chemical treatment: solution in hydrochloric and nitric acids, with subsequent evaporation to dryness, filtration, washing and ignition of the residues.

Oxidised Ores.—Weigh 1 gm. of the ore into a No. 1 beaker, add 15 c.c. hydrochloric acid, cover with a watch glass, and digest at a gentle heat until the ore appears to be quite decomposed, add a few drops of nitric acid, heat until action has ceased, and then wash off the cover with a fine jet of water, and evaporate to dryness. Redissolve in hydrochloric acid and evaporate to dryness a second time to render all the silica insoluble. Redissolve in 10 c.c. hydrochloric acid and 30 c.c. water, filter, transfer all the residue to the filter with a fine jet of cold water, using a 'policeman,' and wash the filter with a little hydrochloric acid and plenty of cold water. Ignite and weigh the residue as 'insoluble silicious residue.'

Complex Ores containing Sulphides.—Treat a weighed portion (0·5 gm.) of the ore with 10 c.c. nitric acid (sp. gr. 1·20) and heat gently until the first intense action ceases and the sulphur has separated out in a clear yellow ball. If the decomposition occurs too violently, due either to too strong acid or too great heat, the sulphur ball will probably include some ore and require to be treated separately. If this occurs it will usually be found better to start a new determination. After the effect of the nitric acid is exhausted and the greater portion of it evaporated off, add 15 c.c. hydrochloric acid and evaporate to dryness; take up with 15 c.c. hydrochloric acid (1—1), boil, filter and wash; evaporate the filtrate to dryness, treat as before, adding the second residue to the first. Wash clean with hydrochloric acid and hot water, ignite and weigh as insoluble silicious residue." H. C. Parmelee.—*Engineering and Mining Journal*, from *Western Chemist and Metallurgist*, Oct. 19, 1907, p. 729. (A. McA. J.)

The Reactions of Calcium Hydride ("Hydrolite").—"Attention has already been drawn to the employment of calcium hydride in the production on a large scale of hydrogen gas suitable for the inflation of balloons. The commercial hydride, known as 'hydrolite,' furnishes hydrogen on treatment with water, and the gas thus obtained is free from acetylene and ammonia. The yield, however, is far short of the theoretical amount, and the slaked lime, which is the other product of the decomposition of the hydride by water, contains an impurity liberating a spontaneously inflammable gas on treatment with hydrochloric acid. This inflammable product has not been definitely identified, for although it behaves like phosphuretted hydrogen, yet the original calcium hydride is apparently free from phosphorus. Should further investigations show that this inflammable gas is set free only in the presence of acid, the decomposition of hydrolite should always be effected with water which has been rendered alkaline, in order that the hydrogen produced may not be contaminated with this dangerous constituent." —*Times Engineering Supplement*, Oct. 9, 1907. (J. A. W.)

The Detection of Traces of Moisture.—"The presence of water, either in the liquid or vaporous condition, plays a very important part in the determination of many chemical changes, and in some instances the reactions are completely inhibited when the reagents are thoroughly dehydrated. In other cases certain preparations are only possible when the reagents can be brought together in anhydrous media. The detection of slight traces of moisture is accordingly of great importance to the chemist. Anhydrous copper sulphate has generally been employed in this test, as the white colour of this salt assumes a bluish tint when the compound is introduced into a moist solvent. In a recent number of the *Berichte der Deutschen Chemischen Gesellschaft*, W. Biltz recommends the use of colourless potassium lead iodide for this purpose, claiming that this double salt is even more sensitive than copper sulphate. A sample of absolute alcohol, which gave no indication of moisture after contact with the latter salt for one hour, showed immediately the presence of water by decomposing the double iodide into its components, potassium and lead iodides, the latter of which was indicated by its intense yellow colour. Bibulous paper and silk fibre may be impregnated with this colourless double salt in a finely divided condition, and these materials then serve as a delicate reagent, assuming a yellow tint in the presence of traces of moisture."—*Times Engineering Supplement*, Oct. 9, 1907. (J. A. W.)

Explosions under Roadways.—The newspapers make frequent reference to somewhat mysterious explosions which occur without any warning in the underground connection-boxes of electric light and power systems. It is usually explained that an accumulation of coal gas, due to leakage from adjacent gas mains, has been accidently fired by a spark from the electric conduits. Thus the electric light companies shift at least half the responsibility upon their special enemies, the gas companies, and explosions continue to occur, to the great discomfort and even danger of the innocent way farer. However, if a recent discovery is to be credited, the electric companies must bear all the blame themselves for these mysterious accidents. Dr. Bassett, speaking before the Society of Chemical Industry, placed on record a case of the production of metallic potassium and sodium by the electric leakage from a subterranean cable of which the insulation had become faulty. The leakage occurred at a point where the negative cable had been joined, and according to Dr. Bassett the electrolysis of the surrounding water containing potassium and sodium salts had resulted in the production of a liquid alloy of the two metals. As everybody knows, either of these two metals will burst into flame if brought into contact with water; and as it is evident that during the electrolytic process considerable quantities of hydrogen must have been formed at the same time, all the materials for an explosion were ready to hand. The inference is that the coal gas mains are innocent in connection with many of these underground explosions. Electrical leakage through moisture must result in the eletrolytic decomposition of the water, and the product must be oxygen and hydrogen gases mixed in the most explosive proportions. Given a connection box or other fitting in which the explosive mixture can accumulate, the smallest spark would precipitate an explosion, and even the spark would be unnecessary if either sodium or potassium in the metallic form had also been produced by electrolysis in even the smallest quantities.—*Chambers' Journal*, Oct., 1907, p. 556. (A. L. E.)

THE USE OF SODIUM CARBONATE AND ZINC OXIDE IN SULPHUR AND ARSENIC DETERMINATIONS.—This is a modification of Eschka's method, and it is stated to have given uniform satisfaction in various assay offices for a number of years. One part dry Na_2CO_3 and 4 pt. ZnO are mixed thoroughly. Half a gram of the sample is then mixed intimately with enough of the $Na_2CO_3 + ZnO$ mixture to afford at least twice as much Na_2CO_3 as would be required by the S, As, etc., present, placed in a small porcelain dish, covered with the reagent and heated to redness for 15 to 20 minutes. The residue is then extracted with water, boiled, filtered, acidified with HCl, precipitated with $BaCl_2$ and weighed as $BaSO_4$. In an As determination the substance is treated as above until the alkaline solution is filtered from the residue. This filtrate is acidified with acetic acid, precipitated with $AgNO_3Aq$, boiled for a few minutes and then filtered. The precipitate of Ag_3AsO_4 is washed thoroughly with hot water, then dissolved in dilute HNO_3 and titrated with KCNS or NH_4CNS, ferric sulphate being used as an indicator. In calculating the As use the proportion 3Ag : 1As : : Wt of Ag found : Wt of As present. The advantages stated are:—(1) The mass resulting from the heating is not fused and can be removed readily from the dish and leached with water. (2) The ease and speed with which sulphides, sulphates and arsenates, etc., are decomposed. (3) No time consuming evaporations are necessary. (4) The absence of a large quantity of alkaline and other salts from the solutions in which $BaSO_4$ and Ag_3AsO_4 are precipitated.—W. C. EBAUGH and C. B. SPRAGUE.—*Journal of the American Chemical Society*, xxix., 10, p. 1475. Oct., 1907, (J. A. W.)

METALLURGY.

NEW AUSTRALIAN CYANIDE PROCESS. — "The recently patented Nicholson apparatus for the filtration of solutions from ores is thus described in a recent issue of the *Queensland Government Mining Journal*:—The slimes are first tipped into a patent agitator with 50% of water, carrying sufficient cyanide salt dissolved in it. The whole is then agitated until the gold in the slimes is dissolved. In order to prove this, the solution is assayed in the ordinary way. The slimes, then in form of a cream, are passed from the agitator into the Nicholson apparatus. This cream goes into the receiver, which is revolving at high speed inside a narrow vessel. The outer vessel travels at a higher speed, and in an opposite direction. The vessel has an enclosed chamber, which contains sand and lime in proportion. On the outside of the wall canvas and such like is fixed. The receiver, while revolving steadily, rises 3 in. from the top, and as it comes down it forces the slimes on the inside of the surface of canvas. The outer vessel travels at a high speed and by the time the receiver reaches the bottom the cyanide solution is forced by pressure through a filter, and runs into a circular launder at the bottom for treatment by zinc shavings. There is a screw on the outside of the receiver, and to this screw movable shovels are fixed by means of springs. As the receiver rises the shovels get caught into the slimes, which are carried out of the vessel at the top into a launder for their reception. A brush surrounds the screw, and above the brush are jets of water. The jets play on the canvas and keep it clean, and at the same time the brush cleans the canvas, and carries everything collected to the top. The apparatus has been constructed in such a manner as to deal successfully with poor slimes that will not pay under present circumstances. A plant to treat 1,000 tons of slimes per week is estimated to cost about £600, while the cost of treatment is placed at about 1s. per ton, and it is claimed that slimes going 12 gr. to the ton would yield a profit. Leading mining men in Adelaide have taken the matter in hand, with a view to forming a syndicate for treating slimes by the new process."—*Mining Reporter*, Oct. 10, 1907, p. 332. (J. Y.)

ALUMINIUM IN STEEL SMELTING.—"Employment of aluminium in metallurgy to prevent blisters and fissures in steel ingots gives excellent results. Suppression of blisters is due to the fact that aluminium has so great an affinity for oxygen that when it is thrown into a crucible of melted steel it absorbs all the oxygen, free or combined with iron, disengaging such heat that the metal is extremely fluid: about 0·01% aluminium suffices."—*Elettricita.*—*London Mining Journal*, Oct. 19. 1907, p. 474. (A. R.)

PROCESS FOR TREATMENT OF COMPLEX ZINCIFEROUS ORES.—"This invention relates to treatment of complex sulphuretted minerals with a zinc base, such, for example, as those of Broken Hill, Australia. The object is to separate the lead and zinc, in the first phase of the process, so as to make the lead insoluble and dissolve all the zinc in form of sulphate. Thus the two metals are brought to a suitable form for subsequent metallurgical treatment. Hitherto great difficulty has been experienced in the treatment of mixed sulphuretted ores, owing to the intimate mixture of the chief elements constituting the ore (blende and galena). It is, in fact, difficult to completely extract the zinc and separate it from lead and other substances. The new process separates the zinc from the lead perfectly by the wet method, each element being obtained in the form of a chemical compound sufficiently pure to make subsequent treatment most easy. The result is attained solely by the wet method without the aid of electricity. The process is as follows: The ore is first reduced to an impalpable powder in one or several suitable mills. It is then heated to boiling in a closed vessel with a ferric sulphate solution containing free sulphuric acid. With properly crushed ore all the zinc enters into solution as sulphate, whilst the lead, silica, and sulphur formed by the reaction remain insoluble. The reaction can be expressed as follows:—

$$ZnS + Fe_2(SO_4)_3 = ZnSO_4 + 2FeSO_4 + S.$$

The solution, free from lead, which remains insoluble in form of sulphate, is filtered in a suitable apparatus; the insoluble residue, after washing and drying, is subjected to dry distillation. The sulphur distils and is collected by condensation, and the residue of distillation is then treated with a view to extraction of lead by ordinary methods. The filtered liquid contains zinc and iron sulphate. Were it left to cool in this state there would be a deposit of crystals of a double salt (ferroso-zincic sulphate). To avoid formation of this double salt the liquid is previously treated with an oxidising agent, preferably nitric acid, so as to transform the ferrous salt, formed during the reaction, into ferric sulphate, very soluble and difficult to crystallise. By allowing the liquid to cool now, only the zinc crystallises in the state of sulphate, and the ferric sulphate remaining in the mother water serves for a new operation. The suitably washed and dried sulphate crystals are subjected to calcination in a muffle furnace. Thus non-volatile zinc oxide is formed, whilst the sulphur is eliminated in form of oxygenated compounds which are retransformed into sulphuric acid by the ordinary methods. This regenerated acid is again utilised, and mixed with new ferric sulphate to treat more ore. The zinc oxide from the muffle furnace is commercially pure, and can be transformed into zinc free from lead by the usual methods.

The process described theoretically needs only, as raw material, mineral coal and fluxes for fusion of the leady residue in a blast furnace. Moreover, the same reagents continually remain, so that the process is cyclic and self-sufficient. Practically, however, as it is impossible to avoid mechanical losses, a certain amount of nitrate of soda must be employed just as in the manufacture of sulphuric acid in lead chambers. The result obtained by this process is almost complete separation of zinc and lead, which enables them to be isolated in the metallic form by ordinary methods without difficulty.

As regards other metals in the mixed ores—*i.e.*, copper, silver, gold—they remain largely insoluble with the plumbic residue, and at fusion of the latter a cupreous matte is obtained, and gold and silver alloyed with lead as usual. If the ore, as is generally the case, contains manganese and iron, all or part of these metals can dissolve in treatment with ferric sulphate. When this is so, the amount of iron in the liquids continually increases, and a time arrives when part of the liquid must be thrown away, which involves loss of zinc, or, the removal of the excess of iron chemically. The most simple method is to precipitate the excess of iron as hydrate or basic ferric sulphate. The liquid is filtered, and the insoluble ferric compound utilised either to prepare iron compounds, or as flux in the fusion of the plumbic residue. As regards manganese, it is dissolved in the form of sulphate, and accumulates in the liquids. As, however, the sulphate of manganese is more soluble cold than hot, it ceases to dissolve at a certain point and remains in the form of sulphate with the plumbous residue. To isolate the manganese the residue is washed with a saturated solution of manganese sulphate from a previous operation to eliminate the mother water, and then the manganese salt is extracted with water. It is then easy to obtain manganese sulphate by evaporation, or oxide by calcination. By operating in this way the zinc sulphate which is deposited from the liquids is pure, and consequently gives an oxide and pure metal without traces of lead, by the ordinary processes in metallurgy."—*London Mining Journal*, from *Revue Chimique*, Oct. 19, 1907, p. 472. (A. R.)

FIXATION OF ATMOSPHERIC NITROGEN.*—"In Italy, a large factory, the first in the world, has been established recently for the production on a large scale of nitrogenised products obtained by the fixation of atmospheric nitrogen. It makes use of a new process discovered by Prof. Frank, of Charlottenburg, and Dr. Caro. The factory is situated at Piano d'Orte, in the province of Chieti. It employs hydro-electric power of 15,000 h.p., drawn from the Pescara river, and it is estimated that in time it will have a daily output of over 13,000 lb. of the new product, which its inventors have called calcium cyanamide. Of course, the process is based upon the property of calcium carbide to fix nitrogen at high temperatures. For reasons of economy, and owing to the immense power at the disposal of the plant at Piano d'Orte, the calcium carbide is here produced on the spot by means of a complete equipment of electric furnaces. The carbide is then ground to a very fine powder by special machinery, and is placed in special iron retorts. It is then heated until it fuses at a temperature of from 800 to 1,000° C. At the same time, very powerful pumps blow over it a continuous current of nitrogen that has been separated from liquified air by the fractional distillation mentioned above. After a couple of hours the carbide is transformed into calcium cyanamide ready for use. Calcium cyanamide as obtained from the retort has the appearance of a very dark mass, composed of extremely fine crystals and of free carbon. Furthermore, it always contains a small quantity of calcium carbide that has not become transformed. Therefore, after grinding the product very finely, it is necessary to expose it for some days to the air, in order that the small quantity of water vapour in the air may remove the portion of calcium carbide still remaining."—*Page's Weekly*, Oct. 5, 1907, p. 832. (A. R.)

ELECTRO METALLURGY IN 1906.—The application of the electric furnace to steel refining is a new development which may lead to very important changes in the iron and steel industries, for in conjunction with gas engines and dynamos it may serve as a means of utilising the enormous power now lost in the waste gases from blast furnaces.

Aluminium can now be autogenously welded, and sheets, rods, tubes of any thickness, can be welded without any difficulty, whilst the points are said to be as strong as the other parts of the metal: this method of welding will probably lead up to increased consumption of the metal in many industries and to its use for larger articles and vessels than have yet been manufactured from it.

Electrolytic methods have been applied with great success on both sides of the Atlantic in the refining of gold and silver bullion, the Moebius process being used for silver and the Wohlwill process for gold. In the Moebius process a dilute solution of silver nitrate containing free nitric acid is employed as electrolyte while in the Wohlevill process a solution of gold chloride is utilised. In America, the Philadelphia and Denver mints are equipped with electrolytic parting apparatus, and a similar installation of electrolytic baths is now being erected at the Government mint, San Francisco. In Europe electrolytic refining is carried on at Frankfurt, by the Deutsche Gold und Silber Scheide-Anstalt, and by the Norddeutsche Raffinerie, at Hamburg, details of the Wohlevill gold refining process having been worked out at the latter refinery. A recent improvement of the Moebius process is the use of gelatine, which gives a smooth coherent in place of a rough crystalline deposit at the cathode.

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* See also this *Journal*, vol. v., p. 247, where *cyanogenamidine* should read *cyano-guanidine.*

Although calcium carbide is being employed chiefly for generating acetylene for illuminating purposes, its application for production of calcium cyanamide is likely to lead to developments of some importance. The use of acetylene gas in the oxy-acetylene blowpipe, for the autogenous welding of metals is another application of considerable industrial importance, since temperatures can be obtained with this apparatus, which approach those of the electric arc, and the size and shape of the flame are more suited for welding purposes. Calcium in the metallic state is one of the latest electro-metallurgical products, the metal being produced by electrolysis of fused calcium chloride and fluoride with a rising cathode, which just touches the surface of the fused electrolyte. This method is adopted to prevent the resolution in the molten electrolyte of the calcium deposited at the cathode. The temperature of the bath is kept at about 670 C., and the process works most satisfactorily with fresh and neutral calcium chloride. The metal is obtained in the form of an irregular rod, made up of a series of buttons, fused together. The metal is dark grey in colour, of specific gravity 1·51. Calcium is now being manufactured upon a commercial scale by the Electrochemische Werke of Bitterfeld, in Germany.

Carborundum is the trade name given to a carbide of silicon, first made by E. G. Acheson, at Niagara Falls, by heating coke, sand and sawdust to a temperature of between 2,000 and 3,000 C. in an electric furnace of the resistance type. The product has the formula SiC, and the manufacture has grown into one of considerable importance on account of the excellent abrasive properties of the carbide.

Another artificial substitute for emery has also appeared, in the form of an electric furnace product called "alundum," obtained by heating bauxite to a high temperature.

Tucker and Lampen have recently carried out some laboratory experiments with carborundum, and have found that the temperature originally given by Acheson for its formation and dissociation are too high. According to Acheson, these temperatures were over 2,500 C., while Tucker and Lampen give 1,600 to 1,900 C. and 2,200 C., respectively.

The electrolytic copper refining industry is the oldest of the electro-metallurgical industies, having been started by James Elkington in South Wales in 1869. The process and methods used by Elkington in this small refinery were similar in all respects to those in use at the present day, copper sulphate being employed as the electrolyte with raw copper anodes and thin sheets of pure copper as cathodes. The only change has been in the magnitude of the operations.

The chief progress of recent years in this industry has been in the substitution of machine for hand labour, the casting of the raw copper anodes and the charging and discharging of the vats by mechanical methods, now being carried out in all the large and up-to-date refineries. The chief improvement on the chemical side of the process has been the addition of a small amount of hydrochloric to the electrolyte in the vats. This, according to Caslon, prevents the loss of silver which otherwise occurs, the insoluble silver chloride being precipitated with the slimes.

Diamantine.—This is a trade name given to a new product obtained by heating alumina with small quantities of silica to a high temperature in the electric furnace. When finely powdered and mixed with clay and water, the new material is said to form a useful wash for the inside lining and walls of furnaces exposed to high temperature. The new product is being manufactured upon a commercial scale by the Diamantine Werke, at Rheinfelden, Germany.

The production of a hard variety of artificial graphite has been carried on since 1892 by Acheson, at Niagara Falls. The method of manufacture is to form first a carbide in the electric furnace, and then to decompose it by increasing the heat up to a point at which it dissociates and the second element is volatilised. Under these conditions the carbon remains in the furnace in the form of graphite. Acheson, in his earlier work, used coke mixed with silica or sand, but he has since found that it is simply necessary to start with ordinary anthracite coal, the impurities of this suffice to provide the second element of the carbide, and when raised to a definite temperature, these elements volatilise and leave the carbon as graphite. The manufacture has been a very successful one, the greater portion of the output being used for electro-chemical and electro-metallurgical work, the Acheson artificial graphite having been found specially suited for electrodes. During 1906 Acheson discovered a process by which the soft variety of graphite can be produced in the electric furnace, and it is expected that this new artificial graphite will become a keen competitor of the natural variety, especially as it shows more uniformity of composition.—*The Engineering Magazine*, Oct., 1907, pp. 105-109. (A. L. E.)

MINING.

TEMPERATURE IN DEEP COLLIERY WORKING.—"Two main, and perhaps more immediate, difficulties appeared to affect this question of the development of deeper mines in the district—one the question of temperature and the other the commercial one. On the former question, much very valuable evidence, some of which was gathered in their own district, was given before the Royal Commission on Coal Supplies of 1902, and it was then stated that coal was being mined at a Lancashire colliery at a depth of 3,483 ft. from the surface with the temperature of the air between 92 and 93 F., no great inconvenience being felt by the men employed. Assuming, as had been variously stated, that 98 was not an impossible temperature for effective labour, it was possible that such a temperature would be reached in the development of deeper mines in North Staffordshire. No practical suggestions had ever been made for the reduction of the underground temperature other than by ventilation, and the question of how far ventilation could reduce the temperature became of serious importance in the deeper mines. Mr. Gerrard communicated in 1904 to the same Commission a very interesting account of an experiment in mining at great depths which had been carried out at the Produits Colliery, Mons, Belgium, where he had observed men working in a temperature gradually rising in the workings to 103° F. in a dry, brisk ventilation in a seam about 3 ft. thick, the depth of the shaft being 3,773 ft. The miners did not suffer distress, nor was there very marked perspiration. The rock temperature was 113°, and it was thought the ventilation had reduced the temperature approximately 20°. Such an experiment, however, hardly reproduced the conditions of working which would obtain in a deep mine with its necessarily large output. In his own experience of working in the North Staffordshire district at a depth of 2,958 ft. probably the deepest workings in the district, the temperature of the seam at a distance of 3,300 ft. from the shaft was 94°, whilst the temperature of the air in the heading was 81¼°, with about 3,000 ft. of air passing. This showed a reduction of temperature of 12¾°. But in a longwall face, upon which 82 men were employed in the same seam at a depth of 2,490 ft., the temperature of the coal was 85°, while the temperature of the air was 82°, at a point where 14,000 cub. ft. of air were passing, showing a reduction of only 3°. It was, he thought, probable that the reduction of temperature which might be obtained by ventilation would in deeper workings be greatly influenced by the quantity of mineral exposed in fresh faces each day. The deeper pits must inevitably have large outputs, and it was difficult to think that the reduction of temperature of 20°, noted in the Produits case, could be general or be obtained in actual work in this country. One conclusion, however, might be drawn from this, that as workings became deeper it would be necessary to have smaller ventilating districts than had been the case in extensive English collieries, so as to reduce the amount of freshly-exposed face swept by each current. Deeper mines would also undoubtedly involve very rigid attention to the question of leakage of the ventilation, and would probably ultimately necessitate that this point should be under the special care of a properly qualified official, to whose requirements many other considerations would have to be subordinate. The instance to which he had referred, of a temperature of 94° of the strata at 2,958 ft., gave a rise of temperature of 1° for each 66 ft. in depth, assuming a constant temperature of 50° F. at a depth of 50 ft. Assuming that a reduction in temperature of 10° by means of ventilation could be obtained, then the temperature of 98° would not have been reached in the atmosphere of the working places until a depth of 3,878 ft. was reached. In the case of the Produits Colliery, however, it was found that with a brisk dry ventilation men were at work in a temperature of 103°, and they might therefore conclude that, so far as the present developments of their local coalfield showed, there would be no overwhelming difficulty with regard to the rise in temperature in working the mines to a depth of 4,000 ft., with their present means of ventilation efficiently carried out."—G. P. HYSLOP.—*Colliery Guardian*, Oct. 18, 1907, p. 715. (A. R.)

RATE OF TEMPERATURE INCREASE WITH DEPTH.—With regard to the main subject of increased temperature, the formerly accepted increase of 1° F. for each 60 ft. of depth is found not to be reliable. With regard to the deep thick coalseam (Warwickshire), we have some very carefully taken data by Dr. J. S. Haldane and Mr. F. G. Meachem. The subject is so fully considered and the conclusions so clear that it is not necessary to say more than that the heat of the rocks will probably not exceed 1° F. for every 120 ft. in depth or half the old estimate.—ALEXANDER SMITH.—*Colliery Guardian*, Oct. 18, 1907, p. 716. (A. R.)

SCIENTIFIC INSTRUMENTS IN MINING.—"The wet and dry bulb thermometers, used for measuring the amount of moisture in the air, are capable of giving very useful information at the present time when so much is heard of coaldust in deep and dry mines, and the methods of dealing with it. We want definite knowledge and comparisons of work and health in dry and humid mines of the same temperatures. Advocates of thorough watering to prevent dust are to be found as well as opponents, and legislation is to be expected on this matter. This legislation should be guided by the widest experience. On this subject some of our shallow mines might give hints, for I could point to more than one case in which the roads for some distance from the shaft have been bricked solely to prevent the decay and fall of the roof and sides under the influence of alternate wetting and drying due to the cold walls of the mine condensing moisture from the air on hot humid summer days.

Some data bearing on this question will be found in the following tables :—

Temperature. F.	Weight in grams of 1 cub. ft. of water vapour.	Percentage increase in the volume of dry air when saturated with moisture.
30	1·97	0·56
40	2·86	0·83
50	4·10	1·21
60	5·77	1·76
70	8·01	2·50
80	10·98	3·52
90	14·85	4·93
100	19·84	6·83

Quantity of water contained in an air-current of 200,000 cub. ft. per minute, saturated with water vapour at

F.	Lb.	
60	164	per minute.
70	228	,,
80	314	,,
90	424	,,
100	567	,,

—Prof. G. R. THOMPSON.—*Colliery Guardian*, Nov. 1, 1907, p. 823. (A. R.)

WIND POWER.—"Although for any purpose requiring a more or less continuous supply of power, the wind is a wholly unsuitable source of energy, there are, nevertheless, many cases in which it can be utilised with advantage. Even if it has to be supplemented by a stand-by such as an oil engine, and worked in conjunction with a storage battery (which is generally an indispensable adjunct), wind-power may prove a source of economy. A few results derived from a series of experiments which have been carried on for some years by the Danish Government may, therefore, be of interest.

The velocities of the wind which are practically utilisable lie between 10 and 50 ft. per second, and the motor must be so constructed as to adapt itself automatically to all conditions, including storms. It has been found that a motor with only four wings is the best, and that if the surface of the wings in square feet is S, the velocity of the wind V in feet per second, and the output in horse-power is W, then $W = SV_3 \div 456{,}000$. Thus for a surface 100 sq. ft., with velocities of 10, 20, 30, and 40 ft., per second, the power available is 0·22, 1·8, 6 and 14 h.p. At the experimental station of Askow, with a petrol-motor as stand-by and a storage battery, an installation of 450 incandescent lamps has been successfully run for two years, at a fair profit even after allowing for interest and sinking fund charges on a 25-year basis."—*Mining Science.*—*Canadian Mining Journal* (Toronto), Oct. 10, 1907, p. 330. (J. Y.)

MISCELLANEOUS.

The Origin of the Gold in the Rand Banket.—"I.—The theory as to the origin of the banket in best agreement with the facts appears to be that which regards the banket as a marine placer in which gold and black sand (magnetite with some titaniferous iron) were laid down in a series of shore deposits. The gold was in minute particles, and it was concentrated by the wash to and fro of the tide, sweeping away the light sand and silt, while the gold collected in the sheltered places between the larger pebbles. The black sand deposited with gold has been converted into pyrites, and at the same time the gold was dissolved and re deposited *in situ*.

II.—The distribution of the gold agrees with that of placer deposits in the following respects:

(*a*) The gold, as in a placer, is contained in the cement and not in the pebbles.

(*b*) The gold has a widespread horizontal, and narrow vertical range.

(*c*) The gold is spread through layers which are conformable to the sediments, and is not deposited in verticals or fault planes across the bedding of the rocks, except in the case of a few unimportant secondary quartz veins.

(*d*) The gold is distributed in patches and not in shoots.

(*e*) The gold tends to occur on the footwall side of the conglomerate beds, or to rest on layers of quartzites which acted as false bottoms in the reef series. Owing to the redistribution of the gold during its solution this rule is not as general as in the case of a recent placer; but it appears to be the general experience through the Rand.

(*f*) The rich patches occur at varying horizons dependent upon the frequent local variations in the currents that necessarily occur during the deposition of a series of deposits upon a shore.

III.—The objections to the infiltration theory include:—

(*a*) The absence of ore shoots.

(*b*) The non-existence of the 'verticals' up which the gold may have been introduced.

(*c*) The limitation of the gold to special seams of conglomerate, and its absence from beds of sand and bastard reef, which lie immediately below rich banket, and must have been equally open to percolating solutions.

IV.—The essential difference in the distribution of the gold between the placer and infiltration theories is that, according to the former, the gold should originally have been deposited at the same time as the deposition of the conglomerates; whereas, according to the infiltration theory, the gold should have been introduced after the formation of the whole sedimentary series. That the gold was contemporary with the conglomerates is shown by:

(*a*) The beds of ore being always parallel to the bedding planes of the rocks.

(*b*) The absence of the 'verticals' of South Dakota and the copper slates of Thuringia.

(*c*) The presence of gold in the conglomerates before they were cut through by the contemporary erosion, which led to the formation of the 'wash out' channel in the May Consolidated mine.

V.—The microscopic evidence shows:—

(*a*) There is no evidence of infiltration, or the presence of the secondary minerals typical of infiltration processes.

(*b*) There has been no conversion of the banket into a continuous sheet of vein quartz.

(*c*) The secondary minerals produced, such as the chloritoid, are typical of pressure-metamorphism, and have been developed alike in rich and barren rocks. Rich and poor banket, bastard reef and quartzite, are shown by the microscopic evidence to have all undergone the same changes, and the richest material sometimes shows less change than barren material.

VI.—The banket differs from the gold ores due to infiltration in other fields—for example, those of South Dakota; and the best general agreement is with that band of modern beach placers which extends for 50 miles along the western coast of the South Island of New Zealand. The Kanowna lead is quoted as a case of the solution and re-disposition of gold in a modern placer.

VII.—The absence of conclusive evidence of any considerable impoverishment in depth is an argument in favour of the alluvial origin of the gold, and is favourable to the further extension of the banket in depth."—Prof. J. W. Gregory, Institute of Mining and Metallurgy.—*London Mining Journal*, Oct. 26, 1907, p. 523. (A. R.)

Cubic Capacity of Coal.—"Nut coal passing through a 1¼ in. screen and over a ¾ in. screen, and having a specific gravity of 1·3 will weigh about 52 lb. per cub. ft., or 38½ cub. ft. per ton of 2,000 lb when loose. If well shaken down, the same coal will weigh about 56 lb. per cub. ft., or about 35½ cub. ft. per ton of 2,000 lb."—*Mines and Minerals*, Oct., 1907, p. 117. (A. McA. J.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

A Manual of Fire Assaying. By Charles Herman Fulton, President and Professor of Metallurgy in the South Dakota School of Mines. Published by the Hill Publishing Co., 505, Pearl Street, New York; 6, Bouverie Street, London, E.C. Price, $2.

Amongst the numerous treatises on assaying this work stands out as a quite refreshing treatment of an old subject. There is nothing, of course, particularly novel in the chapters on furnaces, reagents, sampling, etc., but the author puts in valuable work in those dealing with the reactions taking place in the crucible. Mr. Fulton also treats the assay of complex ores in a sympathetic manner, and the chapter dealing with the platinum ores is in itself worth the price of the book. We are surprised to find that the author quite ignores the various types of magnesia cupels which are so universal on the Rand. Is is somewhat incredible that the satisfaction they give here and in Australia has not caused writers on assaying seriously to consider their qualities, in many ways so superior to boneash. We also find the old heresy of bicarbonate of soda dying hard, and perpetuated once more in this manual. The well-known fact that the extra molecule of carbonic acid is sometimes the occasion of loss to the charge by reason of its rapid evolution before sintering has taken place, should be enough to condemn it, and in view of the many excellent brands of desiccated normal carbonate on the market its continued use seems anomalous. The work contains valuable chapters on errors in gold and silver assaying and the assay of bullion, whilst the dry assay of some of the base metals is also briefly described. The book is printed in large, clear type, with copious references, and is singularly free from typographical errors. (A. W.)

THE ELECTRIC FURNACE IN IRON AND STEEL PRODUCTION. By JOHN B. C. KERSHAW. With 24 Illustrations and a List of Patents. 68 pp. 8½ × 5½. (*The Electrician* Printing and Publishing Co., Ltd.)

"Mr. Kershaw is sufficiently imaginative to picture the coalfields of Europe and America exhausted, so that supplies of iron and steel will depend upon the electric furnace, fed with energy from water resources. This, however, does not hinder him from presenting an altogether practical account of what has been achieved already by such workers as Moissan, Heroult, Keller, Kjellin, Stassano, and others. As this contribution to the subject has already appeared twice in the form of articles in technical journals, it is unnecessary here to do more than call attention to the permanent form in which it can now be procured, and to recommend it as a well-written and well-illustrated account of one of the most valuable processes developed in our time. The first patent is that of Sir William Siemens, dated 1879."—*Times Engineering Supplement*, Oct., 1907. (J. A. W.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 538/08. W. S. Simpson. Improvements in connection with the uniting or welding metals. 31.13.07.

(P.) 1/08. E. M. Weston. Improvements in water supplying attachments for rock drilling machines. 6.1.08.

(P.) 2/08. R. Bowman. An improved process for the reduction of metals from their ores. 6.1.08.

(C.) 3/08. H. Prinder (1), F. Engeler (2). Improvements in ore concentrating tables. 8.1.08.

(C.) 5/08. J. Mazlin. Improvements relating to pick heads usable with removable points, and the like. 8.1.08.

(P.) 6/08. H. Glumann. Trailable point lock. 8.1.08.

(P.) 8/08. P. H. Partridge. Process for seasoning lumber grown in South Africa to prevent it cracking and twisting. 9.1.08.

(P.) 9/08. J. M. Hall (1), T. Kerr (2). A mechanical contrivance for the better extraction of gold or its alloys and black sands, silver and its alloys, and other metals and their alloys. 9.1.08.

(C.) 10/08. W. W. Seay. A process of and apparatus for the production of ice and cold. 10.1.08.

(C.) 11/08. G. M. Capell. Improvements in centrifugal fans and pumps. 10.1.08.

(C.) 12/08. A. F. von Schmidt. Improvements in and relating to the valve gear of rock drills and other tools worked by compressed air. 13.1.08.

(C.) 13/08. P. de Wilde. A process for the extraction of gold from the auro-cyanic solutions. 14.1.08.

(P.) 14/08. W. J. Gold. Improvements in the charging of holes with explosives and means therefor. 14.1.08.

(P.) 15/08. E. Henry. Improvements in the treatment of the surfaces of roads and the like for laying dust and for other purposes. 15.1.08.

(P.) 16/08. H. Schwarz. Improvements in and relating to percussive rock drills. 15.1.08.

(P.) 18/08. Wilhelm Mauss. Improvements in percussion machines. 17.1.08.

(P.) 19/08. James Taylor Carrick. Improvements in treating pyritic copper and nickel ores. 17.1.08.

(P.) 22/08. Donald Mackenzie (1), Richard Nicholson (2), Robert James Nicholson, trading as John Nicholson & Sons. Improvements in hand rock drills and machine rock drill bits. 20.1.08.

(C.) 23/08. James Norman Caught. Improvements in that type of marine and other steam engine in which a reciprocatory motion is converted into a rotary motion. 21.1.08.

(C.) 24/08. Jonas Pehrson. Improvements in hose and pipe couplings. 21.1.08.

(P.) 25/08. Edward Harrison. Improvements in means for building dumps. 23.1.08.

(P.) 26/08. Frank Masters Castleman. Driving gear for forcing grease into bearings or the like. 24.1.08.

(C.) 27/08. Leon Gerard. Improvements in or relating to apparatus for producing and utilising electrical effluvia. 24.1.08.

(P.) 28/08. Alec. Hewitt. Improvements in percussive drills or bits for boring in rock or the like. 25.1.08.

(C.) 29/08. Babcox & Wilcox, Ltd. (1), Theodore Reunert (2), Otto Lenz (3), Harry Reynolds (4), 2 and 3 trading as Reunert & Lenz. Improvements in the furnaces of water-tube or like boilers. 27.1.08.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

COE, W. C., *l/o* Rhodesia; The Bungalow, Hengoed, Cardiff, Wales.
DYKES, J. J., Jun., *to* 11, Royal Exchange Square, Glasgow.
GALBREATH, N. M., *l/o* Roodepoort; 6, Kelly Street, Greenock, Scotland.
HAMILTON, E. M., *l/o* Mexico; Ellesmere, Osterley Park Road, Southhall, Middlesex, England.
JORY, J. V., *to* P. O. Box 109, East Rand.
LACEY, E. D. DE, *to* P. O. Box 206, Roodepoort.
MARQUARD, J. D., *to* P. O. Box 184, Germiston.
MILLER, J., *l/o* S. Australia; Mount Paris Tin Mines, Ringarooma, Tasmania.
MILLIGAN, J. T., *l/o* Pretoria; P. O. Box 647, Johannesburg.
NICKLIN, W., *to* P. O. Box 1908, Johannesburg.
NITCH, H. G., *to* Meyer and Charlton G. M. Co., Ltd., P. O. Box 1127, Johannesburg
PEEBLES, J. G., *to* P. O. Box 32, Cleveland.
WILEY, H., *l/o* Germiston; Crossmolina, County Mayo, Ireland.

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VOL. VIII. FEBRUARY, 1908. No. 8.

Proceedings
AT
Ordinary General Meeting, February 15, 1908.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, February 15th, Prof. J. Yates (President), in the chair. There were also present :—

37 Members : Messrs. R. G. Bevington, W. R. Dowling, A. Richardson, G. O. Smart, A. Whitby, H A. White, Prof. J. A. Wilkinson, W. A. Caldecott, W. Beaver, A. A. Coaton, E. H. Croghan, J. S. Curtis, A. D. Gilmore, G. Goodwin, J. C. Greer, Jr., J. A. Jones, J. Kennedy, G. A. Lawson, H. Lea, J. Lea, A. Lichtenstein, J. P. McKeown, G. Melvill, J. E. Metcalf, J. T. Milligan, S. Newton, T. T. Nichol, F. B. Ogle, W. H. Roe, C. E. Rusden, H. Scarf, R. Stokes, W. H. Stout, H. Warren, F. W. Watson, John Watson and J. Welch.

11 Associates : Messrs. J. Chilton, J. Cronin, C. L Dewar, A. L. Edwards, J. H. Harris, R. W. Leng, C. G. J. Moore, A. Thomas, A. M. Thomas, W. E. Thorpe and L. J. Wilmoth.

6 Visitors and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as published in the *Journal*, were confirmed.

NEW MEMBERS.

Messrs. Croghan and Coaton were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that the candidates for membership had been duly elected, as follows :—

BLAESER, C. A., Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte. Assistant Assayer.

CALDER, WILLIAM LAWRENCE, Transvaal Bischoff Mine. Warmbaths. General Manager.

FILET, GERRIT, Redjang Lebong, Benkoelen, Sumatra, D.E. Indies. Amalgamator.

HAYLLAR, H. F., Simmer & Jack East, Ltd., P. O. Box 47, Germiston. Mine Sampler.

JOHNSON, HERBERT, Simmer & Jack Proprietary Mines, Ltd , P. O. Box 192, Germiston. Shift Boss.

MOSENTHAL, BERTIE, Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte. Assayer. *(Transfer from Associate Roll.)*

The Secretary announced that the following gentlemen had been admitted as Associates by the Council since the last general meeting.

ANDERSON, WILLIAM, Treasury G. M., Ltd., Cleveland. Mine Sampler.

BOWYER, JAMES, East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Mill Foreman.

DAVIDSON, JOHN, P. O. Box 1146, Johannesburg.

DOWLING, ARTHUR EDWARD, East Rand Proprietary Mines. Ltd., P. O. Box 66, East Rand. Millwright.

FOOTE, JUN., ARTHUR B., North Star Mine, Grass Valley, Cal., U.S.A.

HANNA, JOSEPH, Treasnry G. M., Ltd., Cleveland. Cyanider.

HAUTON, GEORGE, Geldenhuis Deep, Ltd., Cleveland Mill Manager.

HOLLAND, GURTH, Simmer and Jack Proprietary Mines, P. O. Box 192, Germiston. Bookkeeper.

SCATTERTY, PETER, Roodepoort Central Deep, Ltd., P. O. Box 114, Roodepoort. Cyanide Manager.

SCOTT, ANDREW, P. O. Box 2361, Johannesburg. Assayer.

STACEY, DOUGLAS WILLIAM, c/o Dr. Vigne, P. O. Box 188, Bulawayo, Rhodesia. Amalgamator.

As a Student :

MILLS, THOMAS LANDALL. The School of Mines, Redruth, Cornwall.

GENERAL BUSINESS.

The President : I beg to move a vote of thanks to Mr. Henry Hay and his staff at the Witwatersrand Deep, for their courtesy and hospitality in connection with our recent visit to that mine.

The motion was then put and carried unanimously.

There is a very sad matter which I would like to refer to. There has just happened in Natal a very serious mining accident, the Glencoe disaster, which has thrown a gloom over the whole sub-continent, having, I am afraid, resulted in the death of about 70 men, many of them belonging to a rescue party. I move that a letter be sent to the Chairman of the Company expressing our regret at the calamity, and the sympathy of this Society with the relatives of the victims.

Mr. A. Richardson: I beg to second the motion. South Africa has, fortunately, been so free from colliery disasters that one like this comes to us with the greater shock. The event is rendered particularly sad because of the loss of one of the rescue parties, the members of which have set before us a noble example of unselfish devotion. Our deepest sympathy goes out to the relatives of those who lost their lives.

The vote was carried in silence, the members standing.

ASSAY OF BATTERY CHIPS AND SCREENS.

By Leslie J. Wilmoth (Associate).

It is some time since the assay of bye-products has been discussed before this Society, and the information to be gathered on the subject is very meagre and incomplete. Having some little time back the necessity to deal with the selling samples of several parcels of bye-products, I was at a loss for suitable methods of assay, and so found it necessary to do a little experimental work to find methods which would yield accurate results. These experiments, I must admit, were few, but the time at my disposal was limited, so I give my experiences with them rather with the object of raising a discussion than of giving anything new. We have lately had some very interesting papers on the routine assaying, but the unusual assays have been to a very great extent neglected by this Society for the past few years, and thus a thorough thrashing out of the assay of these products will not be out of place.

The term bye product has a very wide application and, of course, varies according to the metallurgical work being done in the reduction works. For instance, the bye products of a Rand Reduction Works are very different to those of a copper smelting concern. On the Rand, the chief bye-products of our reduction works may be summed up in the following—now that the Siemens-Halske process has been discarded—battery chips, battery screens, black sands, pan furnace slags, plumbago crucibles, and litharge dross from the pan furnace. The last four of these have been discussed to a slight extent in this Society some few years ago, and although there is room for much experimental work and investigation in the methods applied to their assay, it is my intention this evening to leave them alone and to deal with the assay of the first two:—Battery chips and screens.

Battery Chips.—The usual method of selling these is for a lump sum per ton, irrespective of their assay value. Considering the nature of the material and the difficulty of obtaining a true sample, this seems the best. At the same time, in spite of the difficulty of obtaining a true sample, an assay of the material is of great value in determining the selling price.

Assuming the sample sent into the assay office, the difficulty of an efficient method of assaying it presents itself. It usually consists of lumps of metallic iron varying from the size of a pin's head to as much as half an inch in diameter. The fluxing of metallic iron in a crucible assay, at the best of times, is most difficult, and can only be done when the iron is extremely finely divided. As it is impossible to powder up the lumps, the standard methods of fluxing are useless. The scorification assay is out of the question, as the smallness of the sample operated upon renders the results useless. Again, the removal of lumps of iron by scorification is never very successful. The combination method of dissolving the iron in acid and assaying the residue, I have found by experience to be exceedingly tiresome and slow, to say nothing of being very expensive.

Some little time back, I carried out a series of experiments on the assay of copper ores for their gold contents, and these investigations gave me the idea of the possibility of assaying the battery chips in 1·0 A.T. lots, or even more, by passing the iron into a gold free matte, and collecting the gold in the usual lead button. As these experiments with the copper ore seem to have a direct bearing on the assay of the battery chips, I take the liberty of giving a description of a charge which I found to give most satisfactory results with an 8% copper ore. Not having any gold bearing copper ore to work upon, I mixed an almost pure carbonate of copper ore with a moderately rich sample of banket, so as to give a mixture of 8% metallic copper, and 35 dwt. of gold per short ton. One assay ton of the above mixture contains approximately 2·4 gm. of metallic copper, so I added 1·2 gm. of sulphur with the charge, with the object of passing it into a matte as CuS. The following is the charge I used for fusion:—

Ore	1·0 A.T.
Gran. Soda ...	2·0 ,,
Fused Borax ...	2·0 ,,
Litharge... ...	1·0 ,,
Sulphur	1·2 gm.
Charcoal	1·5 ,,

And a 6 in. iron nail.

It will be seen that litharge is added only in sufficient quantities to give the desired button. The iron nail is added to take up any excess of sulphur that may have been added and so keep the button malleable. On pouring the charge, a matte carrying all the copper will be found on top of the button. The advantage of the above method lies in the fact that, contrary to what might be expected, the matte does not carry any gold. On the particular sample mentioned in several assays, I never found more than an unweighable quantity of gold in the matte, though its silver contents were always high. The above method is not new, but I give the detail to lead up to the application of the method to the assay of the battery chips.

Following on the lines of the above experiments, the use of a matte seems to present an easy solution to the crucible assay of this somewhat awkward sample. I have found that a charge with 50% of sulphur to the chips taken yields a perfectly malleable lead button and a matte carrying practically all the iron originally in the charge taken. As in the assay of copper ore care, has to be taken that no excess of litharge is left in the slag, as any excess of sulphur would yield an exceedingly large button. I have found it advisable to use an iron nail in the case of all assays, using a matte in order to keep the button malleable and the matte free from lead. The following charge yields excellent results with battery chips, irrespective of their size:—

Chips	...	1·0 A.T.
Gran. Soda	...	1·5 ,,
Fused Borax	...	1·5 ,,
Sulphur ...	...	0·5 ,,
Silica ...	...	0·5 ,,
Charcoal ...	...	1·0 gm.

And an iron nail.

It will be seen that silica has been added to protect the crucible, and that an excess of flux has, if anything, been used. After quick fusion, the charge is given a wash down with a small quantity of litharge and charcoal. In this assay the furnace used for fusion must be run at a slightly higher heat than is usual for a gold assay and the crucible *must* be allowed to remain in the furnace a full twenty minutes after quiet fusion. If the heat be insufficient, or the charge be poured too soon, the iron will be found in small lumps on top of the lead button, and not completely absorbed into the matte—the assay being thus rendered useless. I have not, as yet, met with a sample of battery chips assaying more than 6 oz. per ton, and have not found a weighable quantity of gold in the matte of any assay I have done up to the present. In cases where the estimation of the silver is required, it is necessary to save the matte and recover the silver by scorification. With the above material duplicates seldom agree, which is all that can be expected, but at the same time it is astonishing how the average of different batches of assays of the same material correspond.

Battery Screens.—Here again the assayer is faced with an extremely nasty material to sample, as well as to assay. As a rule, these samples reach the office in the form of a number of small clippings of the screens as they have been discarded at the mill. If the sample be small, the pieces can be cut up finer, and a series of, say, half a dozen assays run the same as has been recommended for the battery chips. There is, however, a way of treating the sample preparatory to assay, which has a tendency to yield a far more desirable sample from an assaying point of view. This is as follows:—Weigh the sample as it comes to the office, and then place it in a roasting dish, or a series of roasting dishes, and oxidise in the muffle. A certain amount of care is needed, as with an excessive temperature there is a tendency for the sample to slag as the oxidation nears completion. This oxidation is not very tedious on account of the iron being in the form of a wire. With an ample draught, a little over an hour is quite sufficient. The oxide produced is black, and highly magnetic, and still retains the form of the screening. On completion of the oxidation the roasting dishes are removed, and the contents cooled. When cool, the oxide is weighed and pulverised. It is advisable to take great care in the pulverising of this material as it is often very rich, the gold occasionally being visible as a coating on the mesh. Having pulverised the sample, the assayer is faced with one of two methods of assay, either by direct fusion the same as with an iron ore, or by passing the iron into a matte as has already been described. The most usual method would be a direct fusion, but it would seem that the matte method has a tendency to give higher results on account of the reduced slag losses. We all know that highly basic material tends to carry gold into the slag. By the formation of a matte the baseness of the slag is done away with, and so this tendency disappears. I have noticed that the matte assay usually gives higher results, and I attribute it to the above fact.

These experiments, whilst they certainly point to this tendency in favour of the matte assay, have not proved the point altogether to my satisfaction. The greater number of the assays all show in its favour, but two or three have been very marked in their tendency the other way. These differences however have been so great as to have raised the point as to the effective mixing of the sample before assay. Whilst not positive on the point, the data I have gathered incline me to the belief, that the matte assay gives slightly higher results. The two following sets of figures are averages of two batches of assays done on the same sample at different dates :—

Assay Batch.	Direct Fusion.		Matte Assay.		Difference in favour of matte	
	Au dwt	Ag dwt.	Au dwt.	Ag dwt.	Au dwt.	Ag dwt.
1	175·2	28·0	178·6	14·6	+ 3·4	−13·4
2	173·2	33 6	174·4	16·2	+ 1·2	−17·4

As will be seen the silver losses in the matte are high, but as its recovery presents no difficulty it can be disregarded. In all the above cases the charges were washed down with a mixture of litharge and charcoal just before pouring. Without this final wash the differences are more noticeable. The charges used in the above experiments may be of interest, so I give them :—

DIRECT FUSION.

Oxide	1·0 A.T.
Gran. Soda ...	1·5 „
Fused Borax ...	1·0 „
Litharge	2·0 „
Silica	0·5 „
Charcoal	1·5 gm.

MATTE METHOD.

Oxide	1·0 A.T.
Gran. Soda ...	1·0 „
Fused Borax ...	0·5 „
Litharge	1·0 „
Silica	0·5 „
Sulphur	0·5 „
Charcoal	1·0 gm.

And an iron nail.

This matte assay requires similarly to the assay of the chips a little longer in the fusion furnace than is usual with an ordinary assay. The matte in this case also, as in the assay of chips, may be considered gold free. When the sample is below about 300 dwt. per ton it may be discarded. Above that value it would be wiser to treat it by scorification. With a sample assaying 612·10 dwt. per ton, I recovered the equivalent to 1·2 dwt. from the matte. Glancing back at the assay of these two samples, I think the matte assay gives us a method of considerable accuracy, and one that can be made to fit into the day's work without disturbing the routine of the assay office.

In conclusion, I would like to thank Mr. A. M. Thomas, one of my assistants, for many suggestions and help in getting together the data for these few notes.

The President: I move a vote of thanks to Mr. Wilmoth for his paper, which I am sure will be very useful to many of our assayers.

Mr. A. Whitby: I would like to second the motion. I am sorry I arrived too late to hear the best part of this paper, in which I think it is quite evident there is very interesting matter for assayers to discuss, especially in connection with this point of utilising the matte assay. For myself, I am rather doubtful whether experience will bear out Mr. Wilmoth's findings. Of course, it can only be proved by very careful experiment, and one has to be quite sure before making any definite statements. I think it is a very useful contribution as far as it goes, and all assayers ought to try and follow up this line of research and see if this matte assaying does not give better results sometimes than some of our other methods.

Mr. J. P. McKeown: I would like to ask a question with regard to screenings: Does it make any difference to the assay if the screening is allowed to lie about for some months instead of being dealt with immediately?

The President: Do you suggest that the oxidation of the screenings affects the result?

Mr. J. P. McKeown: I should say from my short experience it makes a difference.

The President: I feel sure that Mr. Wilmoth will attend to that question when he replies to the discussion.

WESTRALIAN WET-CRUSHING PLANTS, WITH SOME NOTES ON LABOUR EFFICIENCY.

By GERARD W. WILLIAMS, F.C.S., A.I.M.M.

The leading companies employing wet crushing in West Australia are the Golden Horseshoe, 150 stamps, the Ivanhoe and Great Fingall with 100 stamps each, the Lake View Consols with 75 stamps, the Oroya-Brownhill with 60, and the Sons of Gwalia with 50 head. In addition to these plants there are a large number of 40, 30, 20 and 10 head mills widely distributed throughout the gold-fields of the state.

Of the mines cited the Fingall is in the Murchison gold-field, the Sons of Gwalia in the Leonora District, and the others are situated on that small belt of highly auriferous country known as the Kalgoorlie "Golden Mile." The Kalgoorlie mills crush "lode matter" which is not so hard as the banket but considerably tougher. The ore may be described as an altered greenstone schist containing about 70% of quartz. The percentage of free quartz increases in depth. The mineral constituents are pyrite, pyrrhotite, and tellurides of gold, silver and mercury. The sulphur contents average 6 %. Tellurides are widely distributed, and in some mines contribute a large percentage of the total gold. Such mines are usually dry crushing, all-roasting plants. Where tellurides occur in wet crushing ores they have to be carefully concentrated, or else the ore must be treated with 'bromo-cyanide.' Normal cyanide solutions are without solvent action on the gold which is combined as a telluride. This fact, together with the presence of much refractory pyritic material compels all the Kalgoorlie wet-crushing plants to resort to close concentration. In the mills enumerated above the battery is simply a primary crusher; secondary crushing is accomplished by means of grinding pans or tube mills, or by a combination of the two agencies.

Of the Kalgoorlie plants mentioned above the Oroya Brownhill, the Lake View Consols and the "new" 100 head mill at the Horseshoe are all-sliming propositions. The coarse sands are crushed fine in pans, and then slimed in tube mills. The Ivanhoe and the "old" 50 head mill at the Horseshoe treat the sands after preliminary crushing in pans, by percolation, and the slimes are treated in the normal manner by agitation and filter-pressing. All these plants are 'continuous process' ones, the ore being in contact with dilute cyanide solutions from the time it enters the mortar boxes.

The tables A and B epitomise the more important data with regard to the larger batteries in this state.

In view of the frequent use of the word slimes in this paper, it might be well to say a few words as to the meaning which this word bears on these fields. The word slimes, as used in Westralia, denotes any product sufficiently finely crushed to yield an economic extraction within twenty four hours with agitation, and which is sufficiently homogeneous to permit of filter pressing. The limit of fineness, therefore, varies with the nature of the ore. On dry crushing and roasting plants the economic limit appears to be attained when 90 % of the product will pass a 100 linear mesh screen. On the wet crushing, telluride containing ores, a much greater degree of fineness is required. The Ivanhoe classes the – 150 product as slimes but on the 'bromo-cyanide' propositions (the Lake View and the Oroya-Brownhill) the limit appears to be – 200.

Throughout this paper I shall use the term slimes to denote the final product as delivered to the agitators. This product is much more granular than the true slimes of the Rand. Decantation methods have been found inapplicable to this class of product, as the fine sands settle out and resist removal from the vats. It is most necessary when contrasting Rand and Westralian methods to bear in mind the different nature of the products known on both fields as "slimes," otherwise serious misapprehensions may arise.

Amalgamation is practised on all wet-crushing plants. In some cases where very coarse battery screening is used, amalgamating tables have been discarded, and all amalgamating is done in the grinding pans. In almost all cases mercury is used in the grinding pans. The practice of inside amalgamation is almost universal throughout West Australia. Where the gold is coarse the practice has certain advantages, but with fine gold the only advantage lies in the fact that mercury being added to all the boxes, the plates require less dressing. But this advantage is

Table "A."

Mine.	No. of Stamps	Monthly Tonnage.	Weight of Stamps. lb.	Duty.	Wear of Shoes and Dies.	Discharge Ins.	Drop.		Screen.	District.
							No. per Min.	Height Ins.		
Ivanhoe ...	100	17,000	1,200	5·5	2·25d. per ton.	2	104	7½	15 × 15	Kalgoorlie
Oroya-Brownhill ...	60	9,000	1,100	6·48	4·26 oz. shoes 4·56 oz. dies	2	108	7¾	10 × 10	Kalgoorlie
Sons of Gwalia ...	50	10,000	1,000	6·68		3½	108	8	20 × 20	Leonora
Lake View Consols	75	11,000	1,200	5·5	4·3 oz. shoes 2·5 oz. dies	2	102	8	300 mesh	Kalgoorlie
Golden Horseshoe	150	24,500	1,270	5·5	2·14 oz. shoes 3·04 oz. dies	3	104	8	15 × 15	Kalgoorlie
Great Fingall ...	100	22,500	1,150	7·0		3	106	8	12 × 12	Murchison

Table "B."

·	Labour and Salaries.	Power.	Repair and Maintenance.	Stores and Sundries.	Totals.	Power Costs per h.p. Day.
	d.	s. d.	d.	d.	s. d.	s. d.
Lake View Consols	4·66	5·64	8·90	3·75	1 10·95	1 4·5
Fingall	5·7	1/2·19	7·47	5·15	2 8·51	1 7
Ivanhoe	4·27	10·31	4·39	5·39	2 0·36	1 4
Sons of Gwalia ...	8·19	1/1·92	5·58	4·49	2 8·18	1 4·5

more than discounted by the time occupied at the monthly clean up, and by the greater opportunities afforded for the adventitious removal of amalgam from the boxes.

On mines where the gold is comparatively coarse and is not intimately associated with the pyrites, simple amalgamation yields excellent results. On the Ida H. gold mine 86·6% of the total gold recovered is obtained by simple amalgamation after crushing through a 500 mesh screen. The ore consists of 70% quartz and 30% of greenstone schist. This mixed ore averages 12% of pyrites.

Concentration.—All the larger wet-crushing plants resort to close concentration. The general method is as follows :—The product from each 5 head of stamps passes over a Wilfley table. The slimes flow direct to the main pump sump and the sands are hydraulically separated into coarse and fine sands, which are in turn fed to coarse and fine grinding pans. The product of the pans passes a row of secondary concentrators. The sands from these tables flow to the main pump sump and are elevated to a series of spitzkasten. The overflow from these passes direct to the slimes settlers, but the sandy underflow is returned to the tube mills (or to the leaching vats, where leaching is practised). From the tube mills the ground ore passes another set of tables and flows back to the main sump.

Where a very coarse battery screen is used (10 × 10) it is usual to run the mill pulp direct to a small spitzkasten and pass the coarse sand through the pans before concentrating.

On the Oroya-Brownhill the battery pulp is projected against a shaking screen which permits the fine sands and mineral to pass through, and retains the coarser product, which is fed to grinding pans before concentration. The advantage of this method over hydraulic separation lies in the fact that the specifically heavy mineral is passed over the concentrating tables before grinding. The difficulty of recovering tellurides is much increased with the fineness to which it is crushed. These classifiers (King's patent) do their work excellently and are especially valuable on this mine, as a large percentage of the total gold is in the form of telluride.

Fine Grinding.—As outlined in the previous paragraph, the mill pulp is passed from time to time over spitzkasten which return the underflow of sandy material to the fine grinders. The secondary grinding of the sands is accomplished by means of grinding pans, of the Forwood-Down type. The final grinding to slimes (– 150 or – 200, as the case may be) is accomplished by means of tube mills, or, as they are usually termed in this State, "grit mills."

Some difference of opinion still exists as to the comparative merits of pans and tube-mills. The tests at the Ivanhoe although carefully carried out and confirmed in general results by subsequent tests at the Oroya have been considered to be somewhat unfair to the tube mill in that the product fed to the mills and grinding pans averaged 23% of 200 product.

But, owing to the nature of the ore, it is not so easy to feed a clean product to the tube mills in Kalgoorlie as it is on the Rand. The net result of these tests was to confirm the superiority of the pan over the tube mill for the purpose of reducing coarse sands to fine, but the superiority of the pans for the purpose of reducing fine sands to – 150 product was less clearly demonstrated. Where all-sliming is practised it is found to be most economical to use pans for grinding the mill pulp, and to pass the product of the pans, freed as far as possible from slimes, to the tube mills. This method is in vogue on the Oroya, the Lake View and the Horseshoe mines.

As an illustration of the capabilities of the grinding pan the following data, obtained in the Ivanhoe tests, is of interest. The battery product containing 56% of + 60 was reduced to a final product containing 95% of – 150 by two pans working in tandem, *i.e.*, the second pan taking the product of the first pan after separation of slimes in a spitzkasten. The two pans coped successfully with the product from 20 head of stamps crushing through a 15 × 15 screen. The same product was fed to a tube mill and reduced to the same degree of fineness. The costs per ton crushed through a 150 linear mesh were 2s. 3·4d. for the pans and 2s. 6·7d. for the tube mill.

The pans used were 5 ft. in diameter, driven at 57 r.p.m., and required 6·5 h.p. The mill was a Krupp 13 ft. × 3 ft. 3 in., driven at 31 r.p.m. and requiring 20 h.p.

The following table shows the normal work done by the 5 ft. pans on the Ivanhoe.

On the Ivanhoe, 10 Wheeler pans, 5 ft. in diameter, driven at 57 r.p.m., each pan requiring 6·5 h.p. Each pan grinds 10 tons per day on sands as shown. About 33% of the mill tonnage passes through these pans.

Table "C."

Screening.	Coarse Sands entering Pans.	Ground Sands leaving Pan.
+ 40	31·4 per cent.	0·6 per cent.
+ 60	21·8 ,,	6·2 ,,
+ 100	17·6 ,,	28·7 ,,
+ 150	6·1 ,,	8·0 ,,
– 150	23·1 ,,	56·5 ,,

The efficiency of the grinding pan depends on several factors, the most important of which are briefly summarised:—

(1) The grinding efficiency is proportional to the circulation. Circulation varies directly as the interspaces between the shoes, and inversely as the decrease in depth of the shoes which is produced by attrition.

Where the shoes are set closely together, *e.g.*, with 1 in. interspaces between the shoes at the periphery of the muller, the drop in circulation, and hence in grinding efficiency, is very marked as the shoes wear down. Consequently the pan with new shoes has to be driven at such an increased rate of speed as shall maintain the normal grinding output with reduced shoes.

(2) For any given arrangement of shoes and dies the grinding efficiency is proportional, within limits, to the expenditure of power.

Coarse sands grind best at high speed, fine sands at medium speed. A small increase in speed involves a large increase in power. At the Perseverance gold mine, where 8 ft. diam. pans are used for grinding roasted ore, it was found that to increase the speed by 14% required an increase of 58% in power.

(3) The grinding efficiency, given equal conditions of circulation, is proportional to the shoe area.

(4) The pans have been found to yield their best results when working on a uniform product; fine and coarse sands should not be crushed in one pan, the coarse sands protect the finer portions from the grinding effect of the shoes. An over load of 10% is usually allowed.

(5) The pulp as usually fed to the pans contains 25% by weight of sands.

Tube Mills.—The difference in the character of the banket and Kalgoorlie ores is very marked in the wear and tear of the mill liners on the two fields.

On the Oroya-Brownhill the tube mills are 13 ft. 7 in. × 3 ft. 8 in., and driven at 32 r.p.m. require 21 h.p. The liners are 1 in. thick and cast locally of chilled cast iron. The wear and tear is uniform throughout the length of the mill and amounts to about 470 lbs. per month. These mills carry a load of 2½ tons of flints. The pulp entering the mill contains about 83% of—150 product, and this is decreased to 76% on leaving. Each mill grinds 27 tons per day through the 150 mesh.

On the Golden Horseshoe and Lake View 16 ft. 4 in. × 4 ft. mills are used. At 29 r.p.m. and consuming 30 h.p. these mills crush about 33 tons of sand per day through a 200 linear mesh.

On the Lake View, manganese-steel liners are used, and the wear and tear averages 250 lbs. per month. The consumption of flints averages 1,000 lbs. On the Golden Horseshoe a set of chilled steel liners, 1 in. thick, lasts about six months. The wear of the liners is equal at both ends of the mill. Silex liners are not used in Kalgoorlie.

The following series of gradings illustrates the nature of the ore treated on the Golden Horseshoe:—

Table "D."

Screening.	Battery Pulp 15 × 15 Screen 22 I.S.W.G.	Feed to Tube Mill.	Discharge from Tube Mill.	Sands as Leached in "Old" Mill.	Final Slime from Filter Presses.
	Per cent.	Per cent	Per cent.	Per cent.	Per cent.
+ 50	27 0	10·0	3·7	6·0	
+ 100	18·0	52·0	46·0	39·0	
+ 150	4·0	9·2	10·8	19·0	
+ 200	1·0	5·5	3·5	7·0	3·0
− 200	50·0	23·3	36·0	29·0	97·0

	Per cent
Percentage extraction on "sands" ...	70·50
,, ,, "slimes" ...	78·02
,, ,, "concentrates"	98·10
Recovery from all sources:—	
By battery amalgamation	27·5
By concentrates (3·7% total tonnage) ...	18·4
By sands and slimes	42·5
(only 2,500 tons of sand treated per month)	
Total	88·4

Average grade of ore, 13 dwt. to 14 dwt.
Monthly tonnage, 22,000 tons.

The tube mill is essentially an impact grinder, whilst the pan grinds by attrition. For this reason the excellent results obtained by grinding pans in Kalgoorlie must be largely discounted when applied to the consideration of banket ores. The wear and tear factor would be largely increased, and on a very hard ore this factor is far more important in pan than in tube mill costs. At first sight it seems as if the pan might be a useful factor in Rand milling practice in that, following Kalgoorlie practice, the pans might be used to break down the coarse product which is at present fed to the tube mill. Although it would be interesting to note the result of running a grinding pan and a tube mill in tandem on banket ores, especially where coarse screening is used in the battery, it seems probable that the increased efficiency of the tube mill would be more than counteracted by the heavy cost of wearing parts in the pan. Certainly the pan deserves a trial at the hands of Rand metallurgists, but in any such trial the pan should be considered as a preliminary fine crusher working on the product normally supplied to the tube mill and delivering the product from the pan, after elimination of slimes, to the tube mill. As a fine grinder, *per se*, the pan cannot possibly compete with the tube mill, but it is possible that working in conjunction, the two might produce a slight diminution in the total costs of fine crushing.

Agitation.—The slimes from the battery and from the subsequent disintegration of the sands pass over the final series of spitzkasten to the settlers. These are mostly pyramidal in construction, and a large area is required. From the settlers the slimes are drawn off to the agitators. The latter are steel vats varying from 12 ft. to 22 ft. in diam., and about 8 ft. in depth. The stirring gear is so arranged that it can be drawn up out of the pulp if, from any mischance, the power is cut off. The stirrers make about 7 r.p.m. The settled slime is made up with solution to a consistency of one part of slimes to one part of solution, and the cyanide strength made up by adding cyanide to 0·06%. The period of agitation varies, but averages twenty-four hours. During agitation the solution strength falls from 0·06 to 0·04%.

On the Oroya-Brownhill and the Lake View Consols the bromo-cyanide process is used. On the Lake View the agitators are 22 ft. in diam. by 8 ft. deep. Each vat holds 60 tons slimes. The cyanide strength of the solution is brought up to 0·075%, and after 15 minutes' agitation 25 lbs. of bromo-cyanide are added. After 12 to 16 hours' agitation lime is added to neutralise the bromine, and the vats are discharged for filter pressing. The agitators in bromo-cyanide plants are covered in. On this plant (the Lake View) the consumption of cyanide, including the cyanide in the bromo-cyanide, averages 1·4 lb. per ton. This amount about represents the average cyanide consumption of the Kalgoorlie wet-crushing mines. The Ivanhoe treats the slimes (53% total tonnage) by bromo-cyanide. The bromo-cyanide is added at the rate of 0·6 lb. per ton of dry slimes.

Filter Pressing.—The large size, 3 ft. 6 in. × 3 ft. 6 in. × 3 in. plates, Dehne type of press, is most commonly used. These presses hold about 5 tons of dry slime. The presses are usually angle closing, but hydraulic closing presses are also used. The filling of a press occupies twenty minutes, washing (at a maximum of 80 lbs. pressure) from ten to twenty minutes, and air drying, usually with air at 50 lbs. pressure, another three to ten minutes. The residues average 15 to 17% of moisture. Filter pressing costs average 1s. 6d. per ton treated. Labour costs form about 50% of the total costs.

A new automatic filter, the Ridgway filter, is now in use on the Great Boulder Proprietary mine. The press is almost automatic in action, one man being able to attend to ten presses. The costs are stated to be much lower than on the older type of press. The Moore press, which like the Ridgway press depends on the action of a vacuum for inducing filtering, has had a successful trial on some out back mines. Vacuum filters appear to have a future before them.

Concentrates. — The concentrates from the Wilfley tables are roasted, mixed with dilute cyanide solution, fine ground in grinding pans with mercury, agitated and filter pressed. The concentrates represent from 3·5 to 5% of the tonnage crushed and contain from 20 to 30% of sulphur. The gold contents represent from 15% to 23% of the total gold.

Sands Treatment.—The methods in vogue here call for little comment. Of the Kalgoorlie mines only the Ivanhoe and the Golden Horseshoe resort to sand treatment, and on the latter mine only a small percentage of the total tonnage is treated in this manner. The residues from the sands plants are higher than the slimes residues, but the costs are less. The all-sliming process, however, gives the best and most economic results. The time of treatment, on 4 dwt. to 5 dwt. ore varies from 5 to 7 days. The Great Fingall, the Gwalia and all the out back mines treat from 55% to 80% of the total tonnage by percolation. Grinding pans are used to break down the coarser sands in the battery pulp.

Precipitation and Clean-up. — On all the larger Westralian mines all solutions are filter-pressed before entering the extractor boxes. As a result of these clarifying presses the solutions enter the boxes crystal clear and the zinc is never fouled by the deposition of siliceous material. Speaking generally, the residual gold values in the solutions leaving the boxes do not compare well with Rand practice; especially is this the case with regard to weak solutions. I think that the explanation is that owing to the nature of the ore the solutions become charged with calcium and magnesium sulphates. The effect of these substances in retarding precipitation is well known. Lead acetate is used as a dipping for the zinc on most plants. On some mines the clean-up is effected with a preliminary acid treatment, but in certain cases the fine zinc and gold slimes are roasted in large C.I. muffles. After thorough roasting the calcined gold slimes are fluxed with 25% of borax and 10% of sand. The calcined slimes contain from 40% to 50% of gold. The resultant bullion contains about 955 gold and silver. This method was introduced by C. Blackett, metallurgist to the Golden Horseshoe, to whom I am indebted for the figures given above. The fluxed slimes are melted in a Martin's tilting furnace. The simplicity of the process, the easy fusibility of the calcined product and the consequent low costs are all in favour of the method, which merits consideration on other fields.

The Continuous Process.—Members are fully acquainted with the continuous process as applied to certain Rand mines. The success or otherwise of the "New Metallurgy." (Why "New"? The process was old before it was introduced on the Rand) does not concern this paper. It is, however, difficult to see what grounds exist for the assumption that Westralian methods can be applied with equal success under entirely alien conditions.

The whole question appears to me to turn on the distribution of the gold contents of the ore. In this respect there is no possible comparison between the two fields. In the banket the gold is almost, if not entirely, confined to the matrix surrounding the pebbles, but in Kalgoorlie it is equally distributed throughout the mass of the ore. On the Kalgoorlie field, where fine crushing to at least a 150 linear mesh screen is absolutely essential for the highest economic result, filter pressing follows as a necessity. Given filter pressing it must be admitted that the continuous process is highly advantageous. The ore is in contact with solution from the time it enters the mortar boxes, and solution of the gold takes place in weak solutions under the most favourable conditions. This materially decreases the time for solution in the agitators. From their nature the true slimes formed from these ores do not lend themselves to the decantation process. Filter pressing, apart from the question of treating the so-called slime (*i.e.*, – 150 sands), is an absolute necessity and the continuous process followed as a logical consequence.

But in view of the low costs and high extractions of the decantation process as evolved on the Rand the introduction of filter pressing seems of doubtful profit. It might be possible to reduce a 0·5 dwt. residue to a 0·3 dwt. residue by all-sliming in approved Kalgoorlie fashion, but unless that all-sliming was accomplished for less than 0·2 dwt. the increased extraction would be of no advantage. The all-sliming plants of this field represent the highest development of this type of process, but their success is due to the fact that they have been evolved as a result of certain conditions, not necessarily the same on other fields.

Labour Efficiency.—So far this paper has been almost entirely descriptive, but I now pass to that portion which will, I think, be not unproductive of controversy. It may be taken as an axiom that the true economic conduct of any industry is very largely a factor of the efficiency of the labour employed. Eighteen months' work on these fields has more than ever convinced me that the Rand does not obtain that labour efficiency which forms so important a factor in determining the low costs ruling in Westralia. As I am dealing at present with reduction plants, all comparisons will be drawn from surface plants. At a future date I hope to lay before you a paper dealing with underground work, and in that paper this subject will be more thoroughly gone into. By the courtesy of Mr. J. Sutherland, general manager of the Golden Horseshoe, I am permitted to append detailed cost sheets for milling as well as the labour distribution sheet for his mine. This mine may be taken, without prejudice to any other wet-crushing plant, as typical of the highest development of this form of metallurgy in Australia. The plant is divided into two portions, an older 50 head mill equipped with concentrating tables and grinding pans, treating part of the ore by agitation and part by percolation in 20 ft. × 7 ft. vats. The more modern mill comprises 100 stamps, 1,270 lb. in weight, concentrating tables, grinding pans and tube mills. The entire product of this mill is slimed. With a few exceptions, such as men employed in the emptying of sand vats and in trucking concentrates who could be replaced by an equal number of natives, I think that you will agree with me that were the Horseshoe transplanted to the Rand that they would employ just as many workers (excepting those previously excepted) and in addition every worker would have one or two boys to help him do his work. It is to this waste of native labour that I would draw attention no less than to the fact that the skilled worker on these fields does his work on his own, whereas on the Rand he almost invariably has the assistance of one or more natives.

This applies to every branch of surface work. On the mine where I am at present, the boiler range consists of five large Lancashire type boilers, four always under steam. This range is fired by one man. In Africa, that man would be a baas, and tell two or more natives what to do. There is an engine room containing the winding engine, ten-drill compressor, mine pump engine and battery engine. One first class engine driver per shift attends to the whole room, the night shift man (there is no hauling on night shift) cleans the engines. How many boys would be employed in this engine room if it were on the Rand?

Of course, it means harder work for the skilled worker, but in view of the scale of wages in vogue on the Rand, it does not seem much to ask that a man should do the same work that he would do here for two-thirds the wage. There is the same waste of labour in the assay offices, in the sampler's room, in the shops, and, to a lesser extent, in the cyanide works. There is no need

THE GOLDEN HORSESHOE ESTATES CO., LTD.—Battery Expenses for Month of July, 1907.

Particulars.	Wages.			Stores.			Total Cost.			Cost per Ton.	
										of 2,000 lb.	of 2,240 lb.
Superintendence	£65	5	4				£65	5	4	·64	·72
Breaking—											
Feeding and tramming	104	14	9				104	14	9	1·03	1·15
Power	72	3	0	£124	1	8	196	4	8	1·92	2·15
Electric light	1	0	1	1	13	2	2	13	3	·03	·03
Spare parts				73	16	10	73	16	10	·72	·81
Repairs and maintenance	56	16	11	52	9	6	109	6	5	1·10	1·20
Conveying—											
Attendance	105	9	3				105	9	3	1·03	1·15
Power	5	14	10	23	9	6	29	4	4	·28	·32
Electric light	1	5	1	2	1	4	3	6	5	·04	·04
Repairs and maintenance	33	3	8	65	0	10	98	4	6	·96	1·08
Milling—											
Millmen	139	2	7				139	2	7	1·40	1·52
Feeding	99	13	4	4	2	7	103	15	11	·92	1·14
Power	243	18	7	667	18	8	911	17	3	8·92	9·99
Screens, shoes and dies				95	18	6	95	18	6	·94	1·05
Mercury, 594 lb.				60	2	10	60	2	10	·59	·66
Water, 1,200,000 gal.	23	1	6	301	11	1	324	12	7	3·18	3·56
Electric light	4	15	6	7	17	3	12	12	9	·12	·14
Assaying and sampling	12	17	0	9	6	10	22	3	10	·20	·24
Smelting and retorting	13	7	11	13	3	8	26	11	7	·23	·29
Repairs and maintenance	179	0	1	304	8	9	483	8	10	4·72	5·29
Classifying and Concentrating—											
Attendance	130	2	1				130	2	1	1·27	1·42
Taking to bins	48	6	0				48	6	0	·47	·53
Power	31	10	10	76	4	7	107	15	5	1·05	1·18
Electric light	1	17	9	3	2	0	4	19	9	·05	·05
Repairs and maintenance	100	15	2	47	16	9	148	11	11	1·45	1·63
Elevating—											
Power	85	18	2	247	15	11	333	14	1	3·26	3·65
Electric light	1	2	9	1	17	3	3	0	0	·03	·03
Tailings, wheel attend. and repairs	35	16	9	19	5	5	55	2	2	·53	·60
Tailings, pump ,, ,,	39	14	9	53	18	9	93	13	6	·92	1·03
Cornish lift ,, ,,	14	8	11	20	6	6	34	15	5	·34	·38
Return Water—											
Attendance	58	2	6				58	2	6	·57	·64
Power	21	9	9	109	9	8	130	19	5	1·28	1·43
Electric light		12	7	1	0	8	1	13	3	·02	·02
Repairs and maintenance	47	12	8	60	10	10	108	3	6	1·09	1·19
Fine Grinding (Grit Mills)—											
Attendance	46	12	6				46	12	6	·45	·51
Power	264	8	1	749	0	10	1,013	8	11	9·81	11·10
Electric light		17	8	1	8	11	2	6	7	·02	·02
Liners				250	0	0	250	0	0	2·45	2·74
Flints (10 tons 17 cwt. 2 qt)				48	11	6	48	11	6	·47	·53
Repairs and maintenance	31	11	5	44	8	11	76	0	4	·74	·83
Grinding Pans—											
Attendance	47	11	9				47	11	9	·46	·52
Power	72	19	9	162	7	4	235	7	1	2·30	2·58
Electric light	2	5	3	3	14	6	5	19	9	·06	·07
Shoes and dies				126	0	0	126	0	0	1·23	1·38
Repairs and maintenance	8	16	8	3	19	4	12	16	0	·12	·14
Total	£2,254	3	2	£3,838	2	8	£6,092	5	10	4/11·6	5/6·73

Tonnage treated, 21,910 of 2,240 lbs.
=24,539 tons of 2,000 lbs.

Cost per ton of 2,240 lbs., Wages 2/0·69, Stores 3/6·04, Total 5/6·73.
,, 2,000 ,, 1/10·05 ,, 3/1·53 ,, 4/11·58

for me to point out to you the advantages to be gained by taking away superfluous natives, and employing them on useful work, whilst leaving the white worker to do his own work instead of, too often, watching some one else do it. I freely admit that I prefer a native to make up assay fires, to fetch and carry generally about the plant and to do these little jobs which otherwise I must needs do myself. At present with the thermometer at 110° in the shade I wish for him even more. But economically the practice is indefensible.

The function of the native worker is to perform unskilled work, not to relieve the white worker of legitimate exertion. So long as natives were plentiful the practice of allowing surface workers the services of one or more natives was the natural outcome of the false labour conditions obtaining in South Africa. The growing need for an increased supply of coloured labour in its legitimate sphere, as true unskilled workers doing work which it is not feasible to give to whites, renders it necessary that every practicable economy in the employment of such labour should be effected.

I venture to think that the figures quoted in this paper afford conclusive proof that it is possible to effect such a reduction. An indirect advantage which would be secured by reducing the number of surface natives, numerically few in any one case but, taken over the whole Rand, comprising a large number, would be the increased efficiency of the white workers. This will prove a gain not only to the cost sheet of the mine but to South Africa as a whole.

I have not drawn attention to these points in any spirit of captious criticism. I believe that every industrial centre can learn something from other centres, and as the predominant characteristic of the Westralian gold-fields is a high labour efficiency, both on the surface and underground, it appears most necessary to draw especial attention to this factor. This is the one great lesson which the Australian gold-fields can teach the Rand.

LABOUR DISTRIBUTION.

Golden Horseshoe G. M., Kalgoorlie. Average Tonnage, 22,500 Short Tons.

	Men per 24 hours (3 shifts).	Totals.
No. 1 Mill (50 head)—		
Shift bosses	3	
On 12 concentrator tables and 13 grinding pans ...	3	
Battery feeders	3	
Stamp man	1	
Casual	1	11
No. 2 Mill (100 head)—		
Shift bosses	3	
On classification	3	
On 30 concentrator tables and 10 tube mills	3	
Battery feeders	3	
On trucking concentrates ...	3	
On tailings elevators ...	3	
Stamp men	2	20
Sands Plant—		
On pumping solution ...	1	
On discharging vats ...	4	5
Slimes Collection and Agitation Vats—		
Shift bosses	3	
On pumping solution ...	3	
On settlers	12	18
Filter Press Plant—		
On filling and washing ...	3	
On discharging and closing...	28	
Casual	2	33
Disposal of Residues—		
On sands	4	
On Slimes	6	10
General Foremen (day and night)	2	2
Total ...		101

Total Treatment Costs—

	s.	d.
Labour and salaries ...	2	5·89
Power	4	6·02
Repair and maintenance ...	1	8·58
Stores and sundries ...	4	2·21
Total ...	12	10·70

The President: As most of you know, Mr. Williams was on these fields for several years. He often writes to the Council, and it is really very gratifying to see the interest he continues to take in the Society. In connection with this particular paper his remarks on labour efficiency have been of especial interest to me. He has a happy way of expressing his views and I am sure you have been just as much interested in them as myself. I move a vote of thanks to him for his paper.

Mr. J. S. Curtis: I am an old hand at pan amalgamation. It has been largely discarded in America, and I think it has also been discarded in Europe, but may exist still in Australia for all I know. As regards fine grinding, the modern methods utilise tube mills or other crushing mills, which are far in advance of the pans, because the wear and tear of the shoes and dies of the latter is extremely great.

THE LABORATORY: ITS ECONOMIC VALUE.

(*Read at October Meeting, 1907.*)

By A. McArthur Johnston, M.A., F.C.S. (Member).

DISCUSSION.

Mr. A. Whitby: The subject of this paper is of great interest, and the author has dealt with it in such a comprehensive manner that at first sight there appears little to offer in the way of criticism.

At the outset, however, I find that he considers the status of the assayer to be vastly improved. If increasing the quantity of work without extending the facilities for doing it is a measure of the enhanced value placed on the assayer's services I have nothing to say. It is true that somewhat more reliance is placed on that portion of the assay work which comes under metallurgical cognisance, and it is equally true that more accuracy is required, but at the same time the quantity of work demanded has been greatly increased, as I said before. When the chief assayer has practically to do nothing but weighing from early morn to dewy eve, as must follow if he does 150 to 200 assays a day, I fail to see how supervision of the other work can be efficiently carried on. The ancient controversy as to what an assayer could really do in a day and what was actually demanded of him is wrapped up in this question. Multiplication of assistants does not fill the bill when the equipment is merely sufficient to properly cope with half the amount of work. Nor do I see that the assayer is in any way benefited by the assumed changed conditions. The department still remains prejudiced by the attitude of the authorities in considering it as a necessary evil, the one which shows no tangible fruit, and therefore the last to be considered when improvements are wanted.

Coming now to the main object of this paper, which is, as I take it, to impress on the mining authorities the economies effected by, and the consequent necessity for a proper examination of, their stores, there cannot be two opinions, and the argument holds good that a suitably-equipped laboratory must justify its existence, either directly or indirectly. On the other hand, it is very much like butting at a brick wall to attempt to assail the present system of buying, and in contemplation of the subject one is further faced with an undeniable reluctance on the part of merchants to submit to any scrutiny of their goods. That this is an iniquitous state of affairs is quite obvious, and it must in time give place to more reasonable methods. The system of buying on guaranteed samples taken from bulk which finds acceptance in all advanced communities does not yet appear to have been adopted in this country, although we sometimes pride ourselves on being up to date. It was in operation in the firm of manufacturing chemists to which I was attached 25 years ago, and the rule was absolute that no drug, chemical or article used in the works could be passed into stock without a rigorous examination by the analytical department. I merely mention this to show that, although the opinions expressed by the author and Prof. Wilkinson indicate that the system is not universal in England, yet there are firms who fully appreciate the advantages of a works' chemist, even if they do not always show their appreciation tangibly.

It is now ancient history, but as an instance of the failure, even of metallurgists, to grasp an important issue, I must refer once again to the attitude adopted in the matter of cyanide. It took the metallurgical world several years to grasp the fact, of which the chemist was ready to assure them, that cyanide was in no sense pure with a 98% guarantee. In one or two instances, rather than admit the error, metallurgists advanced several quite puerile objections to the use of sodium cyanide, and continued to advocate the use of the potassium salt, regardless of what was quite obvious to the chemist and a standing source of amusement to the manufacturer that they were only being supplied with an impure sodium salt. To my own knowledge (and admittedly that of the manufacturer) some of the cyanide sold here contained upwards of 20% of common salt to bring it up to the 98% standard! Whilst this was the required basis, no effort was taken in the manufacture to ensure purity in the substances used, and even as much as 25% of inert or deleterious matter found its way into the manufactured article. I leave it to our business men to work out the economy or otherwise here involved.

The materials mentioned by Mr. Johnston as requiring the attention of the chemist are, of course, merely examples, and do not cover the ground, and it is one of the characteristics of a well appreciated laboratory, that it is continually finding fresh material to work on. I might here instance a matter of considerable importance as affecting the health of the mine employee. It is no secret in the trade that owing to excessive competition certain medicaments used in the mine hospitals are not what they should be. Take, for example, carbolic oil and disinfectants. With regard to the former, it rarely comes up to the standard laid down by the pharmacopœia, and the latter may have the odour of the phenols without containing enough to justify their use. Again, it is customary to supply, usually under contract, cough, diarrhœa, tonic and other mixtures made up in quantity to prescription. The effect of this is sometimes to get what used to be quite common in the Dutch medicine trade, where the "wonder essentz" was often made the dumping ground for any derelict tincture or unknown preparation in the chemist's shop. Remembering that it is extremely difficult to analyse a complex prescription, it would seem to be the better plan to order the ingredients separately; then it might be possible to judge of their quality, which afterwards would be extremely difficult.

The author mentions somewhere in his paper that his viscosity results convey very little to the engineer. Surely this is an anomalous state of affairs, probably arising from the variety of instruments in use for this purpose. Archbutt and Deeley have proposed to reduce all results to what they call absolute viscosity, so that the engineer is made independent of the instrument used, whether Redwood, Engler, Saybolt or Doolittle torsion. The present practice of reporting the rate of flow in terms of a particular instrument for such an important test is to be condemned. Here I would wish to remark that it is highly desirable to adopt standards in apparatus and methods for the technical examination of all substances used in the arts. At present the section of applied chemistry associated with engineering is in a lamentable state of confusion, and the public analyst is frequently forced by the works' chemist, who is generally in a better position for it, to acquire apparatus which is excessively costly, and at the same time merely transitory in advantage. There is little doubt in my mind that the old Thomson calorimeter, for example, was capable in the hands of an experienced worker of giving results quite conformable with the work done by any ordinary class of coal, whilst admittedly failing with anthracites and cokes, but this does not form any justification for the compulsory usage which has been, as I say, forced on us of an instrument which costs nearly ten times the price and does not give any better results when worked against extended experience with the Thomson. More important, however, is uniformity in methods of analysis. Taking steel for example, several recognised methods are in use for the determination of carbon, most of which are based on the solution of the iron and the subsequent oxidation of the carbon to CO_2 which is either weighed or measured. It is manifest that the initial process of solution is one on which all chemists should be agreed. Quoting from Stillman, the American committee on standard methods of iron analyses found that, contrary to the usual practice, the solvent at the outset must not be neutral, but strongly acid. As they say, this is quite contrary to the English practice, and if my own small experience with high carbon steels is to be relied on, not by any means a safe procedure, the evolution of hydrocarbons being admitted as a source of loss. I do not wish to take up time unnecessarily, but this solitary example could be multiplied to cover almost every article which comes under the notice of the engineering chemist. Even amongst public analysts the insufficiency of methods for examining butter forms only one example amongst many.

Mr. E. H. Croghan: In contributing these few remarks on Mr. Johnston's practical paper, I feel sure he deserves our best thanks, since through the medium of our *Journal*, "The Laboratory: Its Economic Value," is brought before the public eye, and especially before the various mining groups. With the exception of the group Mr. Johnston is connected with, the rest of the groups in the past and up to the present have only dabbled with the subject and are still in the mire in failing to appreciate its value and possibilities.

The laboratory is an important asset, and certainly deserves their combined serious consideration. By this I mean it should not be necessary that each group should go to the great expense of having a properly constituted laboratory of their own, but should rather contribute to the maintenance of one central laboratory, with whose co-operation it should be possible to work in perfect harmony to the mutual benefit of the whole industry. In my opinion, it is a narrow-minded policy to adopt the attitude that if one group secures better results than another, why should it give the benefits of its experience to another? I feel sure that investigations have been instituted, say, by a group, when the questions have already been solved by others, thus incurring unnecessary expenditure.

To a certain extent we are not all here for the benefit of our health, still, while we are here, we should work as much as possible not only for ourselves, but also for the mutual welfare of the country and posterity.

It would be a very poor outlook for any country depending, say, on its agricultural resources if farmers did not co-operate, exchanging ideas, and also verifying suggestions made to them by their Central Government Agricultural Department. It, therefore, seems to me that, with the Technical Societies we have here and the institution of a combined group laboratory, we should be in a position not only to help in developing the gold industry to its maximum value, but also to devote time to the best treatment of low-grade propositions, refractory ores, etc., as well as the conduct of other base metal industries, which we all hope will be opened up in the near future. The expense and up-keep of such a laboratory need not necessarily devolve solely on the shoulders of the mining groups, but should also be granted State aid. It would certainly be better to capitalise a realisable asset such as the preceding, rather than institute so-called experimental funds. The benefits that must accrue from its proper conduct should go a long way to secure the confidence of European and other investors, and in conjunction with the development of the agricultural industry should do a good deal in helping to form the foundations of what we all hope will be a prosperous country. In making the preceding remarks as a private individual, I quite appreciate that I am advocating what may seem a policy possibly dangerous to private laboratories, but if a scheme should ever be entertained, which to me seems only a matter of time, I feel convinced that the matter can be and should be arranged without infringing their interests and means of livelihood. I now come back to one or two details of Mr. Johnston's paper from a chemical point of view. At present the method of reporting cyanide as containing over 100% potassium cyanide must seem a curious point to the ordinary eye. Would it not be better to report in terms of cyanogen, at the same time including, if necessary its equivalent in terms of either potassium cyanide or sodium cyanide, or both? The meaning would then be at least clear to any person.

I do not quite agree with the method of determining volatile matter in coals adopted by Mr. Johnston. The standard method is, I believe, carried out in a platinum crucible and lid, only continuing the heating one minute after the disappearance of the flame and done over a good Bunsen burner. At one time I employed Mr. Johnston's method, but I found it was difficult to get uniform results with muffle heating. In regard to the platinum crucible method I found that, with different sizes of crucibles, results varying by 0·5 to 2% were obtained on a basis of 30% volatile matter being present. It, therefore, seems evident that a standard size of crucible should be used for comparative data. In regard to a factor for ascertaining the evaporative power of coals from their proximate analysis, I take the liberty of quoting one given to me by Mr. H. D. Bell, assayer, York mine :—

If the moisture and ash are carefully determined, and the difference between the sum of their percentages and 100 be multiplied by 0·14, it will be found that with ordinary bituminous coals, the results agree very well with the Mahler bomb data. As regards its application to anthracite coals, I am not in a position to state if it is satisfactory.

The subject of boiler feed waters and their treatment seems to me a matter on which many engineers require some education. Processes are often adopted, the mechanical arrangement of which is not only primitive, but is often more expensive than or costs as much as the installation of a proper plant; also the chemicals recommended and the quantity and mode of application are not conducive to the wellbeing and life of the boilers. With these crude plants the oils from the condensers are never eliminated as they should be, for even if they are purely mineral, they should not be allowed to re-enter the boilers. On the matter of the purification of boiler feed waters and boiler corrosion, I have some interesting data, which I hope to be able to place before you at some future time. The use of boiler fluids, anti-incrustants, etc., should never be allowed, especially if recommended to be employed inside the boilers. If engineers only knew how they are imposed upon by these wonderful cures, I feel sure they would leave them severely alone. At any rate, if desired, they should never be adopted without ascertaining their absolute composition, and even then should be handled with suspicion.

With reference to lubricating oils, greases, and rope dressings, I think only mineral oils should be employed, especially for high pressures and temperatures, and no admixture of vegetable or animal oils of any description should be allowed, or if present they should be brought under certain guaranteed percentages. In conclusion, I am sure that the publication by Mr. Johnston at some future period of the results of the microscopic analysis of metals will afford a paper of great local practical value.

The President: I am sorry to say that Mr. Johnston is very ill at present, and I am afraid that it will be some time before he is about again. It is hoped, however, that he will be well enough to reply to the discussion at the next meeting.

THE ORIGIN OF THE GOLD IN BANKET.

(Having reference to Prof. J. W. Gregory's paper on the same subject, read before the Institution of Mining and Metallurgy.)

(*Read at January Meeting, 1908.*)

By J. S. Curtis (Member).

DISCUSSION.

Mr. J. T. Milligan: On p. 201 of the *Journal* the author states: "It must be remembered that the matrix of the banket is basic, and being basic, quartz cannot predominate." In support of the author's infiltration theory I have noticed that where a considerable number of quartz veins or stringers are found running through the banket, these sections of reef are invariably of poor value. No doubt, at these points, owing to the predominance of quartz the reef is acidic and precipitation did not take place or was much less than the average. These quartz veins being, of course, filled in subsequent to the deposit of the banket, it follows that they could have had no influence on gold already deposited in a placer.

SOME FEATURES OF SILVER ORE TREATMENT IN MEXICO.

(Read at January Meeting, 1908.)

By W. A Caldecott, B.A., F.C.S. (Member).

Part I.—Works Practice.

The President: Mr. Caldecott brought this Cuchara to-night for inspection. It is used for finishing "pannings" in Mexico, and it is really identical with the horn spoon which we occasionally use in prospecting. Of course, the idea is that you can do your final panning much better in such a spoon than in an ordinary pan. You can get a very nice tail indeed which shows up far better on the horn background than in the ordinary pan.

THE INCIDENCE OF METHODS OF PAYMENT ON THE EFFICENCY OF MINERS.

(Read at November Meeting, 1907.)

By Kenneth Austin, M.Am.I.M.E.

DISCUSSION.

Mr. A. Richardson: The author, in his short paper, has touched the fringe of a subject which gave rise to much controversy during the strike and to many vigorous, if biassed, statements before the Mining Industry Commission. It has also been the object of much close investigation in America, the country which leads in the scientific solution of labour problems, and one of their ablest experts recently laid it down as a fundamental principle that high wages and low labour costs form the foundation of the best management.

That a perfect system will be devised, in this generation, at any rate, seems doubtful, but as the science of mining becomes more exact so do the prospects of finding a suitable system become brighter. Turning to the various points raised by the author as they occur, I am, first of all, in full agreement with him as to the undesirability of introducing a Compulsory Arbitration Act, as its enforcement in other colonies has given rise to grave constitutional difficulties.

In New Zealand, for instance, the Labour Conference declared that it had no confidence in its Arbitration Court, and the Federation of Employers stated that as a result of the Act there had never been greater friction in labour matters than at present, and that, while the cost of production had been seriously affected, the conditions of the workers as a whole had not improved. In that country, because of the business dislocation and unrestfulness begotten of the Act and the destruction of all sympathy between master and man, it is often asked whether it would not be better to have a strike and have done with it?

Again, in New South Wales the coal miners ever since the introduction of the State Industrial Arbitration Act have generally refused to acknowledge any judgment with which they were not in sympathy. A compulsory arbitration act is, to put it plainly, merely socialism masquerading in a fancy dress, which in these days should deceive no one.

The right of workers to form labour unions for mutual protection is admitted, but where they injure the country and themselves is in fixing a maximum daily task which no man must exceed, thus stifling individual effort and reducing everything to a dead level of wooden performance.

The argument that the miner does not accomplish the best results because of the insecurity consequent upon the rule of 24 hours' notice of dismissal seems scarcely impartial, since in the other departments where it is in force no such prejudicial effect has been observed

Coming to the merits and demerits of the usual systems I notice that day's pay has none of the former granted to it, a verdict which seems somewhat harsh, as a system that confers a large measure of contentment due to a fixed wage, lessens friction by reason of the few opportunities for disputes, and encourages a friendly co-operation throughout, must have some commercial value. With it mining is more carefully and safely performed, though there is generally a waste of explosives, and it has in its favour a simplicity of accounts which is not, as the author would have us believe, a feature of the contract system.

If, however, as on some mines a close account is kept of fathomage, footage, and stores, in order to check and locate inefficiency, then the accounts are probably not so simple as those of contracts. The bonus system I consider to be more elastic than the contract system. Its accounts are not at all complicated when once understood, and dissatisfaction amongst the men is reduced almost to the day's pay standard. Whatever happens, a wage is assured, there being no possibility of coming out in debt or of those violent fluctuations in pay common to the contract system which are the cause of so much suspicion in the minds of the workers. The contract system, is a more fruitful source of discontent amongst the men than any other. It demands a great deal of supervision to check the inevitable sloppy mining, due to a chase for fathomage at any cost, careful mining as regards the following of reefs, and keeping stope widths low. Bringing down dangerous hanging or supporting bad ground with a plentiful supply of pillars. is looked upon by the contractor as a matter where compulsion, not initiative, is required: moreover, the incentive to do his best work is effectively restrained by the fear of a "cut."

A man on day's pay or bonus is as contented as ordinary human nature admits of and readily obeys instructions, but directly he goes on contract he will scarcely slide down a stope unless he is given a price per foot for doing so, and his attitude towards those in authority is more aggressive than agreeable. This sort of thing, seemingly inseparable from the system, renders the society of the miners and officials mutually distasteful. largely destroying the good understanding so helpful to co-operative effort.

If contracts are given out for three or more months, and I am in favour of this, the agreement requires to be very carefully drawn up, as the miner is a bird of passage and can flit when small cheques threaten, whereas the company must sit and be shot at by the big cheque. A 10% retention would, I think, be sufficient to cover all ordinary contingencies, and these contracts should include drill maintenance (exclusive of hoses), for which 2s. extra per fathom would probably suffice. Spare columns should be provided, if asked for, and wherever possible one man should be given one face. I see no reason why the letting of faces by tender should not be given a trial, those wishing to tender being allowed to inspect the working place and to familiarise themselves with the conditions. This is often done in Australia and America and might solve some difficulties here.

If any new system is to be introduced it is advisable to make the change very gradually, as miners always regard any new scheme with great suspicion, and it is of vital consequence that the complete sympathy of those in control underground be secured, as without their loyal support important details may not receive proper attention. It is, to mention one point often overlooked, a help to the efficient carrying out of any system of control, if shift-bosses are given staff quarters and a staff position. In a mine as in an army it is detrimental to discipline if a lieutenant is on too familiar terms with his men.

As the author has not—more's the pity—given us any concrete examples to facilitate criticism, I will furnish a few brief details of some systems that have been tried here.

A system analogous to the payment of colliers by the weight of coal produced was employed at the City and Suburban after the war up to the time of the strike, and, it is said, gave satisfaction. Every truck was tallied and credited to the miner from whose stope it came, payment being based on the number of trucks sent to the shaft. To check the inclusion of waste a bonus was given for close stoping widths, a pro rata charge was made against the men for waste sorted out at the surface, and 1s. a ton was given for waste sorted out below.

Another one, originated by Carl A. Davis, of the Centre Star and War Eagle mines of Rossland, B.C., known as the hole contract system was as follows:—

There were two machine drilling shifts, 7 a.m. to 4 p.m., and 4 p.m. to 1 a.m., with one hour meal interval in each. The location of holes was indicated by the shift bosses, misplaced holes being disallowed for payment. A special blasting gang worked from 1 a.m. to 7 a.m., and handled all the explosives. The advantages claimed were:— Saving in explosives owing to their being used by picked men, and blasting on one shift induced better ventilation, especially applicable to stopes with irregular ore bodies which present great difficulties to measurement. Machines of one man could be moved about at will or mixed with those of another without mixing measurements The men increased their earnings from \$3·50 to \$4·0—\$4·25 per shift, while the ton stoped now cost the company \$0·477 as against \$0·865 with the wage system, and development \$8·79 per foot as against \$11·14 with an average speed per month of 97·8 ft. with the hole contract, as against 50·8 ft. with the wage system.

This scheme was tried at one of our deep levels for about a year, but as it has been abandoned one must conclude that it did not come up the level claimed for it by its originator. The price paid was 6½d. per foot on the first rig up, and 7d. on the second; the men averaged about 25s. a shift, and disputes were few; the holes drilled on the day shift were measured by the night shift bosses and so on, all misplaced holes being disallowed. The objections to this method appear to have been that it threw a great amount of extra work on the shift bosses and the men were up to every conceivable dodge to get holes measured twice over, they also so strenuously opposed the introduction of the blasting gang, the most important feature, that it had to be abandoned, with a consequent waste of explosives. This somewhat novel method, which has been subjected to much adverse criticism, might have been a success had the conditions been more favourable, and it is worthy of notice that the Waihi Grand Junction mine of New Zealand has recently discarded the contract system in its favour.

The next is a bonus system, as follows:— Machine stoping, rate, 45s. per fathom, all pillars being measured in. Supplies, blasting gelatine 58s. a case, gelignite 45s., native shifts 2s. 9d. each, other stores at cost. Each man was guaranteed 27s. a shift in consideration of his running as many machines as efficiency permitted: half the balance, after the wage had been charged against the fathomage profit, went to the miner, all debits being wiped off by the company. Hand stoping, rate, 62s. to 65s., according to stope, all pillars measured in, supplies as before; 16s. 8d. per shift was guaranteed and charged against the fathomage profit, half the balance going to the miner, the debits being wiped off. Hand development 32s. 6d. per foot, the conditions being similar. Machine development, a contract at 27s. 6d. per foot for the first 80 ft., and a bonus of 5s. per foot for anything beyond. The next is a sliding scale system of bonus for hand stoping, one which gave satisfaction on the mine in which it was employed. The stopes were divided into five classes, according to the difficulties they presented, and payment was made on the basis of the average fathomage broken per 3 ft. hole, a reasonable consumption of explosives being permitted for each class of stope.

HAND STOPING.

Fathoms per 3 ft. Hole.	Classes and Prices per Shift.				
	1	2	3	4	5
·038					10 0
·042				10 0	12 6
·046			10 0	12 6	15 0
·050		10 0	12 6	15 0	17 6
·054	10 0	12 6	15 0	17 6	20 0
·058	12 6	15 0	17 6	20 0	22 6
·062	15 0	17 6	20 0	22 6	25 0
·066	17 6	20 0	22 6	25 0	27 6
·070	20 0	22 6	25 0	27 6	
·074	22 6	25 0	27 6		
·078	25 0	27 6			
·082	27 6				
Explosive allowance in fathoms, per case	4·8	4·6	4·4	4·2	4·0

The miner was debited with half the cost of any excess of explosive used over these allowances and credited with half the value of the amount of explosive saved. The maximum wage allowed was 27s. 6d. per shift, the minimum wage being 10s. per shift and a discharge note.

Machine stoping on contract in the same mine: A flat rate of 45s. per fathom, all pillars being given in as fathomage. Blasting gelatine, the explosive used, was issued at 53s. 9d. a case. Native shifts 2s. 9d. each: other usual supplies at cost price. Spare columns were provided to facilitate quick rigging up, but they were taken but little advantage of.

Finally, there is the tribute system where payment is made per pennyweight on a sliding scale, according to the value of the rock, a charge of 10s. per ton being made on all rock crushed as an additional check on the inclusion of waste.

These few examples, to which many others might be added with advantage, will serve to show that it is not for want of trying that the Rand has not yet evolved a standard system for the payment of miners, and will support the view that all new ideas are worthy of a trial, as it is only by making experiments and taking risks that the industry can move forward.

The meeting then closed.

Visit to the Witwatersrand Deep, Ltd.

On Saturday, 25th January, a large number of the members visited the Witwatersrand Deep, Ltd., on the invitation of the General Manager, Mr. Henry Hay. The whole of the surface works was inspected, and several points of especial interest were given close attention, among these being the Templar skips and cages, the new coal burning crucible and muffle furnace combined, and the method of utilising waste heat for warming the solutions of the slimes plant, a paper on this having been read quite recently before the Society by Mr. A. Salkinson.* The following figures and descriptions are of interest:—

Templar Skips and Cages.—The east shaft has been equipped in four compartments with Templar skips and cages. The Templar skip is well adapted for compound shafts, running on guides in the vertical, and on wheels in the incline portion of the shaft.

In the two western compartments the two skips are replaced at change of shift by two three-deck cages. Each cage holds 36 persons; 12 persons on each deck. A quick changing device has been installed by which the two skips can be taken off and the two three-deck cages put on in 15 minutes. A three-deck ladder platform, travelling on rails, is brought up to the shaft when the cages are in use, so that all the decks of the cage can be loaded or unloaded without shifting the position of cages in the shaft.

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* The Utilisation of Waste Heat in Slimes Settlement, by Mr. A. Salkinson, this *Journal* vol. viii., June, 1907, pp. 403-406.

Coal-burning Crucible and Muffle Furnaces Combined for Assaying. — Cost of erection (approximate), £53. Estimated capacity, 500 assays per day.

N.B.—Calculated on a basis of 2 A.T. and 3 A.T. charges fused in G and J crucibles, producing 27 to 30 gm. lead buttons, cupelled on No. 7 cupels.

Estimated total cost of fuel per assay when fusing and cupelling 500 assays per day, 0·46d.

Average number of fusions per month, done in this office	3,500
Previous cost of fuel per month (coke)	£63 0 0
Present cost of fuel per month (coal)	16 0 0
Saving per month in fuel alone ...	47 0 0
Life of muffle in coke furnace ...	One week
Cost of muffles per month ...	£3 12 0
Life of muffle in present coal furnace	12 weeks
Cost of muffle per month ...	£0 6 0
Total saving per month	50 6 0
Total saving per year	603 0 0

Other advantages.—All roasting and igniting done in third muffle without delaying cupellation. All parting, evaporating solution, assays and distilling water done on hot plate by waste heat. Greater control of temperature and air supply to muffles. No coke or gas-carbon to break. No choking fumes of SO_2 and much less heat for the man who does the fire-work.

Heating of the Solutions in the Slimes Plant.—The following figures are based on 10,000 tons of slimes treated per month :—

Size of present plant	4 collectors	40′ × 11′ 4″
	6 treatment tanks	42′ 6″ × 14′ 4″
	2 treatment tanks	40′ × 14′ 4″

Total capacity (collectors and treatment tanks) 214,240 cub. ft. or 646 cub. ft. per ton of slimes treated per 24 hours.

Treatment tanks capacity, 157,989 cub. ft. or 474 cub. ft. per ton of slimes treated per 24 hours.

Summer temperature of solutions (without heat) about 70° F.

Summer temperature of solutions (with waste heat as at present) about 85° F.

In winter (without heat) solution temperature is about 50° to 55° F.

Results with Warm Solutions.—September to November, 1907.

Average Slimes Tonnage.	Average Charge Value.	Average Residue Value.	Average Extraction.
10,091	1·979 dwt.	·320 dwt	83 83%

The above figures do not include gold recovered from the slimes dam ; adding this, the following results are obtained :—

Average Residue Value.	Average Extraction.
·237 dwt.	88·02%

To treat 10,000 tons of slimes, according to the average Rand practice, five extra tanks would be required at a cost of £5,000, giving a total capacity of 915 cub. ft. per ton of slimes treated per 24 hours. The installation of the heating scheme cost £500. At present 11,700 tons of slimes are being treated per month.

At the close of the inspection the party was hospitably entertained. The President (Prof. J. Yates) moved a vote of thanks to the management, which was responded to by Mr. H. Hay.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

ANTIMONY AND TIN ASSAY.—" One-half gram of the alloy is dissolved at 40° to 50° in a mixture of 15 c.c. of nitric acid (sp. gr. 1·42), 15 c.c. water, and 6 gm. of tartaric acid. When dissolved, the liquid is heated to boiling, and the tin is completely precipitated by adding a slight excess of phosphoric acid (sp. gr. 1·3). The liquid is now diluted with boiling water to 300 c.c. and the tin phosphate is collected and washed with a hot solution of ammonium nitrate. The filtrate containing the antimony is neutralised with ammonia mixed with excess of ammonium sulphide and then acidified with acetic acid. The antimonious sulphide is finally converted into the peroxide by treating with fuming nitric acid with the usual precautions. The tin phosphate is dissolved in warm ammonium sulphide, precipitated as sulphide by acidifying the cold strongly diluted solution with dilute sulphuric acid, and is finally converted into oxide by nitric acid in the usual manner. Other metals, if present (such as copper and lead), will remain insoluble in the ammonium sulphide, and can be further tested."—*Australian Mining Standard*, Oct. 30, 1907, p. 411. (A. McA. J.)

DETERMINATION OF CO IN ATMOSPHERIC AIR.*—All satisfactory methods for estimating small amounts of CO are based on the reaction

$$I_2O_5 + 5CO = 5CO_2 + I_2$$

which was first observed by Ditte (*Bull. Soc. Chim.*, 13, 318, 1870). Every c.c. of CO at N.T.P. liberates 0·00227 gm. iodine and 1 c.c. of CO_2. This method was used by the authors to determine the amount of CO in compressed air, but considerable difficulty was experienced because of the continuous apparently spontaneous decomposition of the I_2O_5 even below 100°. This was found to be due to the lubricant lanoline used. Finally the method recommended by Kinnicutt and Sanford (*Jour. Amer. Chem. Soc.*, 22, 14, 1900), was used with the addition that the CO_2 passing from the KI solution is absorbed by standard $Ba(OH)_2Aq$, thus acting as a check on the iodine titration using $N/1,000$ $Na_2S_2O_3Aq$. By placing another $Ba(OH)_2$ tube before the one containing the I_2O_5, both the free CO_2 and that produced by the free CO can be determined in one operation by two titrations with the same reagent. — *Journal of the American Chemical Society*, xxix., 11, 1589, Nov., 1907. (J. A. W.)

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* See this *Journal*, vol. vi., Oct., 1905, p. 134, Feb., 1906, p. 254 ; and vol. vii,, July, 1906, p. 14.

NEW RADIO-ACTIVE ELEMENT. "The *American Journal of Science* gives details of the discovery by Prof. B. B. Boltwood, of Yale University, of a new source of radium. Evidence has been obtained of the existence in the uranium metals of a new radio-active element which emits both alpha and beta radiations, which produces no emanation, and which resembles thorium in its chemical properties. It is, without doubt, a disintegration product of uranium and is in all probability the immediate parent of radium. The name 'ionium,' derived from 'ion,' is proposed for this new substance, which would seem to be appropriate, because of the ionising action which it possesses, in common with the other elements which emit alpha rays."—*Times Engineering Supplement*, Oct. 23, 1907. (J. A. W.)

WATER PURIFICATION.—"Permanganate of lime is quite suitable for purification of water, owing to its destructive action on organic matter. In this respect it excels permanganate of potash and even corrosive sublimate, and, of course, is preferable to the latter, being neither poisonous nor corrosive. It suffices to add a centigramme to one litre of water to ensure complete sterilisation in five minutes, as this salt is decomposed in the presence of organic matter, into manganese dioxide oxygen, and lime salts. The flavour of the water is not affected."—*Mining Journal* from *Mouvement Industrielle*, Nov. 2, 1907, p. 538. (A. McA. J.)

TEST FOR TITANIUM.—"In the case of silicates and other minerals, Mr. G. Gallo proposes the following process for determining the presence of titanium: The precipitate obtained with ammonia, generally consisting of hydrates of aluminium, iron, zirconium and orthotitanic acid, is treated by boiling with excess of potassium hydrate solution to separate the alumina. The aluminium hydrate is precipitated with ammonia in the solution thus obtained after acidification with nitric acid, and then weighed. The residue from treatment with potassium hydrate, containing iron, titanium, zirconium, is dissolved in hydrochloric acid, and again precipitated with ammonia; after careful washing, it is desiccated and weighed. Thus, one has Fe_2O_3, TiO_2, ZrO_2. The residue is then fused with potassium bisulphate, and the sulphuric solution of the fused mass allowed to cool. The iron and titanium are then determined, as usual, by volumetric analysis. By differences the zirconium is then estimated.—*Gazetta Chimica Italiana*, Sept. 20.—*London Mining Journal*, Nov. 16, 1907, p. 601. (A. R.)

METALLURGY.

PROPERTIES OF CAST IRON.—"The author reports the results of an exceedingly careful test to determine the variation of properties of cast iron with its composition. He selected five kinds of pig iron of the following compositions:—

	Si.	Mn.	S.	C.	Combined C.	% of C in Combination
I.	2·52	0·89	0·022	3·56	0·30	8·4
II.	1·96	0·64	0·019	3·64	0·39	10·7
III.	1·50	0·59	0·065	3·42	0·55	16·0
IV.	1·12	0·64	0 071	3·32	0·60	18·0
V.	1·08	0·72	0 054	3·26	0·99	30·3

Each of these was melted carefully in a crucible and cast into bars in well-dried sand moulds, 30 cm. long and 2, 3, 5, 10 and 15 cm. in diameter. These bars were allowed to cool completely in the moulds, then broken and sampled by planing off the whole of the fractured surfaces. The same chemical analyses were made as above, and are compared with the analysis of the pig iron used to note the effect of the casting into small or large bars, with the consequent rapid or slow setting and cooling, respectively. The comparison of the five analyses of the pig irons given above show the effect of decreasing silicon or increasing sulphur or both, on the state of combination of the carbon. Comparing each pig iron with the bars cast from it, the bar 10 cm. in diameter showed practically the same proportion of the carbon as combined carbon as in the pig iron, the smaller bars a two to three times greater proportion of combined carbon, and the larger bar a slightly less. If we assume that the strength increases as the proportion of combined carbon increases, we are led to the conclusion that for thick-walled castings a sulphur content of 0·15 is decidedly advantageous in increasing the proportion of combined carbon.

Next, test bars were cast of a series of eight pig irons in which Mn, S and total C were kept practically constant, but silicon was decreased from 2·34 to 0·97, while sulphur simultaneously increased from 0·064 to 0·146. The percentage of the total carbon present in the combined state rose from 13·7 up to 26·4, and the mechanical tests showed a steady increase of tensile strength until Si 1·20 and S 0·116, and a steady decrease of bending strength. A similar set of eight tests were made with silicon, phosphorus and sulphur constant, total carbon slightly increasing (3·27 to 3·62%), and manganese rising from 0·35 to 1·56. These tests showed this increase of manganese to be practically without effect on the state of the carbon, but to reduce sharply the tensile strength and the bending strength after it exceeds 1%. A similar set of eight tests was made, keeping Si, Mn, S and total C constant, but increasing phosphorus regularly from 0·24 to 1·43. The result showed this increase to have practically no effect upon the state of combination of the carbon, but to decrease regularly the tensile strength and the bending property.

The author concludes that if a casting is to have high tensile strength and stiffness, it should contain 20 to 25% of its total carbon as combined carbon, which is attained by it containing 1 to 1·5% of silicon and 0·06 to 0·15 sulphur, according to the thickness of the casting, with Mn about 0·5 and phosphorus 0·2 to 0·5. For high bending strength with low tensile strength, 1·4 to 2·0% silicon, with Mn, P and S as low as possible. For high strength of both kinds choose the mean composition between those indicated."—MAX ORTHEY.—*Electro-chemical and Metallurgical Industry* from *Metallurgy*, 1907, vol. iv., p. 196; Oct., 1907, p. 420. (A. McA. J.)

ELECTROLYTIC PRODUCTION OF PURE TIN.—Pure tin cathodes several centimetres thick, constituted by a perfectly compact adhering metal as fine as electrocopper cathodes, have been currently obtained, and under industrial conditions. In this form the metal is merchantable without being cast into ingots. More than this, cathode tin, owing to its extreme purity, will no doubt soon be of considerably greater value than ingot tin.

Investigations made on this subject have been kept very secret but the patent protecting them has been published nearly a year and a somewhat extensive plant has been at work for several years. The material treated is not tin scrap but electrolytic sludge, which is deposited in the vats where old bronze in sulphuric liquor is electrolysed to extract copper. The process, however, is applicable also for treatment of crude ores after mechanical preparation, oxidising, roasting and leaching, or of any product which gives stannate of soda by the wet or igneous method, such as tin slag, waste from tinning, old tin plate. If the product treated consists of sludge from sulphuric electrolysis of bronze, nearly all the tin is in form of stannous hydrate, which must previously be transformed into stannic hydrate. The oxidation, made before washing the sludge, is effected naturally and rapidly at the atmospheric temperature by simply drying in the air. As tor the impurities, they may be, according to the origin of the raw material, hydroxides, oxides, sulphates, arsenites, arsenates, antimonites, antimoniates, etc., of common metals like copper, lead, iron, bismuth, etc. In the case of bronze there may be very appreciable traces of silver, which might make it profitable to utilise the last remnants, which are not abundant, from the operation, which contain all the precious metals. If the product treated consists of remnants from tinning, tin slag, etc., it will contain iron, lead and copper chiefly. The process comprises three chief operations, viz.: (1) formation of a stannate of soda solution; (2) purification of this solution: (3) electrolysis."—Copied by *Mining Reporter* from *Transactions London Mining Journal*, Sept. 19, 1907, p. 271. (J. Y.)

FORMATION OF GRAPHITE IN CAST IRON.—"In the *Comptes Rendus de l'Académie des Sciences* for September 23, 1907, the author sums up the results of certain experiments on cast irons containing varying quantities of other metals. His results indicate that nickel has a characteristic action on the carbon present, the effect of which is plainly shown by the micrographic examination of the alloy. As the proportion of nickel increases the perlite disappears and sorbite makes its appearance, then the cementite assumes to some extent the acicular form rarely seen in ordinary cast iron. Further addition of nickel leads to the formation of γ iron and needles of troostosorbite, and then the sorbite disappears. The γ iron still increases, and the needles of troostosorbite, which often contain a nucleus of cementite, diminish until finally only γ iron and graphite are discernible, although certain parts of the iron in the vicinity of graphite show traces of cementite. A comparative study of cast iron containing manganese showed that in no case was γ iron produced, sorbite being generally formed. The manganese hinders the precipitation of graphite, in this respect behaving like chromium, which also forms a double carbide with cementite. Aluminium and silicon resemble nickel in their action on cast iron and promote the formation of graphite. This investigator is extending his experiments to cast irons containing vanadium, tungsten and molybdenum."—L. GUILLET.—*Times Engineering Supplement*, Oct. 30, 1907. (J. A. W.)

CYANIDE PROCESS.—"In the treatment of gold and silver bearing ore in which a large percentage of copper is present, some difficulty is found in the recovery of the precious metals, particularly of the silver, owing to the fact that the cyanide solution becomes heavily charged with copper. It takes up 40% of its own weight in copper, and when the solution is once loaded with copper, no silver values can be obtained. I. Anderson (861,628, July 30, 1907) proposes the following treatment in successive steps: Pulp containing precious metals and copper is first treated with potassium cyanide, the extracted metals are precipitated with sulphuric acid. The sulphuric acid is then neutralised with lime, whereby the cyanide solution is regenerated. The pulp is then again subjected to treatment with the regenerated solution, whereby the rest of the precious metal values is taken out. This solution now containing gold and silver is then used for the treatment of fresh pulp to take out the copper and so on as before."—*Electro-Chemical and Metallurgical Industry*, Oct., 1907, p. 425. (A. McA. J.)

STEEL HARDENING.—"The discrepancies in the literature of hardening are due to the fact that certain essential practical features have not met with recognition. Thus any hardened steel, previous to hardening, must have had a considerable amount of mechanical work put on it. Any mechanical stresses present must be relieved by annealing. If the properties of a hardened tool depended solely on the production of a certain type of structure, then cast tools would answer. Experience shows that the fullest properties are only reached on material which has been worked and annealed. Recognition of these features and a study of advanced hardening practice would result in the removal of many discrepancies and tend to elevate metallography into a science from which practical men could draw inspiration in times of difficulty."—P. LONGMUIR.—*Iron and Steel Institute* in *Electro-Chemical and Metallurgical Industry*, Oct., 1907, p. 397. (A. McA. J.)

SHORT ELECTROLYTIC COPPER DETERMINATION.—"I present a few notes and observations made during an 18-months' practical test of the Guess three-hour electrolytic method for copper as modified by R. W. Senger and myself. The tests were made in the smelter laboratory of the Cananea Consolidated Copper Co.

Apparatus.—The laboratory is equipped with 320 electrolytic units of the Guess-Haultain design. The terminals are of $\frac{5}{16}$ in. cast aluminium with a cathode and two anodes in one piece. The 3 in. poles project 2 in. through a board and below each set is a slide to support the beaker. Each unit has two vertically corrugated anodes of 0·001 in. platinum foil 0·5 cm. wide by 12·5 cm. long, and one corrugated cathode with a blade 6·25 cm. long by 4 cm. wide, the handle of which is of the same piece, 0·7 cm. wide and 6·25 cm. long. These electrodes are fitted into slots cut in the terminals and held by contact pressure. Long service has proved that cathodes of 0·0025 in. foil weighing about 3 gm., containing just enough iridium to make them springy, are far more serviceable than any other. Hollow aluminium plugs may be dropped between any two terminals to short circuit the assay.

The switchboard in the smelter laboratory controls 200 assays in banks of 60, 70 and 70. The current is taken from 250 volt direct current mains, and led through two 80 ohm rheostats in series, for each cabinet. The rheostats are of such resistance that the current may be cut down to 1·50 amp. with one assay on each cabinet and with sufficient steps to keep the current at 1·50 amp. while the cabinet is being filled. A 350 ohm coil resistance may be thrown in on No. 1 cabinet to reduce the current to 0·4 amp. for 16-hour assays.

Procedure for Ores.—Weigh 0·5 gm. ore (1 gm. for low-grade ores) into a 100 c.c. tall beaker, with vertical sides; add 5 c.c. nitric acid and 1 to 1·50 c.c. sulphuric acid; boil gently to fumes; bake dry to remove any sulphur cakes; cool; add 5 c.c. nitric acid and 20 c.c. water, and boil gently 5 minutes. Add 1 c.c. nitro compound, fill with cool water, allow to settle, and electrolyse with 1·50 amp. for 3 hours.

For simple ores and concentrator products, weigh 0·5 to 2 gm., as necessary, into the 100 c.c. beaker;

add 7 to 8 c.c. nitric acid; boil gently with the beaker covered until free from nitrous fumes, leaving 5 to 6 c.c. free acid. Add 1 c.c. nitro compound, fill with cool water, allow to settle and electrolyse as above.

The nitro compound is prepared by digesting 25 gm. No. 4 hard oil of the Standard Oil Company, in 250 c.c. nitric acid on a warm plate of 12 hours. Dilute with cold water to 500 c.c., mix thoroughly and filter. The action of the nitro compound is to retard the sudden deposition of copper when the current is first passed through the solution, and to allow a slow, constant deposition giving a hard, clean deposit. It prevents the deposition of the small amounts of antimony contained in our ores. If the compound is too concentrated, a scum may form about the top of the electrode when the copper depositing about it will leave a dark brown streak which cannot be removed and gives too high a result. On slags and low grade tailings the nitro compound is of little advantage.

The Electrolysis.—On standing 5 minutes all 0·5 gm. charges are sufficiently settled to begin electrolysis. Two-gram charges take longer and fine slimes may not settle entirely. The latter may be filtered or the copper redeposited from the first cathode on a clean cathode for absolute results, but for ordinary work the first deposit is sufficiently accurate, the increase being but a few hundredths of 1 .

Before starting electrolysis the anodes must be set firmly in the slots and at equal distances from the cathodes; otherwise the deposition will be uneven and violent evolution of gas at the cathode will occur, giving a dark deposit. On starting electrolysis, evolution of gas at the cathode is common and occurs less frequently during and at the end of the operation. This may be stopped by an instantaneous short circuit of the assay.

It has been proved that in the 3 hours with a 5 to 6% acid solution, the deposition of 0·025 gm. copper is absolutely certain and that 0·050 gm. can be deposited leaving but a faint trace of copper in solution. This gives the limit of a 10% Cu on 0·5 gm. ore. Beyond this point it is advisable to allow the assay to remain one-half to 1 hour longer according to the copper contents, adding 1 c.c. nitric acid and 0·5 c.c. nitro compound if it is found that the deposit turns dark or the solution becomes too greatly heated. The discrepancy from the theoretical is due to the fact that the theoretical deposition is from a saturated solution, and such conditions never occur in practice.

Care of Electrodes.—At the end of electrolysis, the cathodes are removed either by lifting out of the beaker or by lowering the beaker, washed thoroughly, dipped in alcohol, dried and placed on velvet trays. The deposits are hard and firm, eliminating any danger of removal of copper by handling. The electrodes are cleaned by dipping in warm nitric acid, washing thoroughly and allowing to stand, handles covered, in boiling dilute caustic soda. With this treatment, the weight changes 0·0001 to 0·0002 gm. in 10 days.

This method, under conditions described, is entirely satisfactory and removes as much as possible the personal equation so vital to all methods. One man can handle 200 assays in a day and have sufficient time to do other work about the laboratory."—PERSIFOR G. SPILSBURY.—*Engineering and Mining Journal*, Oct. 26, 1907, p. 773. (A. R.)

MINING.

THE FLOTTMAN DRILL.—"The Flottman drill is of the piston type, with solid borer, and weighs 15 kilogrammes. It is 520 millimetres long, 125 in the wider part, and gives 2,000 strokes per minute. In ordinary work across the bed and similar operations, one man suffices. In raising for normal or oblique boreholes two workmen are needed. It takes four to six minutes to bore a hole 1·50 metres deep in soft schist and soft sandstone, nine to eleven in hard sandstone, and eighteen to twenty in dense conglomerate. These drills are difficult to work in humid rock. The dust produces a viscous slime which prevents rotation of the borer, in spite of a strong jet of water. However, their structure is simple, and they require less repairs than drills on carriages. They are particularly useful in soft and moderately hard rock. Finally, they are easy to handle."—*Bulletin de l'Industrie Minerale*, 3e livraison, 1907.—*London Mining Journal*, Oct. 26, 1907, p. 509. (A. R.)

PROSPECTING WITH CHURN DRILLS.—"Believing that the great horizontal ore bodies of the Ely district could be more cheaply and expeditiously tested by the use of the drill rather than through the usual tedious and expensive method of shaft-sinking, the Ely Central Copper Company purchased two drills, manufactured by the Keystone Drill Company, of Beaver Falls, Pennsylvania. These drills were hauled from Cherry Creek to the mines, a distance of sixty miles.

These machines are type No. 3, and weigh 12,000 lb. each. A complete outfit of tools and equipment for recovering any parts of the drills lost in boring the holes were included in the order.

The first drill was installed on the property in August, and we had a hole completed to a depth of 308 ft. within twenty-three days, although but one shift was engaged in this work. It is far more economical in wood and water to keep the drills running continuously with three shifts. More than three times the sinking can be done in the same number of days, as steaming up in the morning generally comes out of the one shift. The drill averaged 13·3 ft. per shift for the twenty-three shifts, but if shut-downs are deducted the drill averaged 19·6 ft. per shift of actual operation.

We have found in all our work that casing is necessary in every hole. At times we have sunk 200 ft. without casing, but to go deeper casing was necessary, which meant that the hole had to be rimmed out at an expense of more than the original cost of the hole for the 200 ft. I caution any one contemplating the purchase of a drill to provide 7⅝ casing for one-third the depth contemplated, 5⅝ casing for two-thirds the depth, and 4¼ casing for full depth. This we had not done, and probably 25% has been added to the cost of each hole, working without sufficient casing. Trouble may not be encountered in several hundred feet, but a soft stratum of 10 ft. will require the casing for the entire hole.

We soon learned that the wear and tear on the drilling cable was severe. Under ordinary circumstances each rope is safe for 1,500 ft. of drilling. When we appreciated that our ropes were deteriorating, an order was placed at once. By the 1st October the cable had not arrived, and we were compelled to order by express from Ohio a coil of rope weighing 1,850 lb. With each drill two ropes of the same length as the depth of the hole contemplated should be ordered.

Water is required for both the boiler and for diluting the drillings so that the sand-pump may bale them. The greater quantity is required for the boiler. This item of expense is local with each hole

It is advisable to haul water in a good wagon-tank, as the drills may be moved many times, and the cost of pipe might be greater than the cost of hauling. A drill-man and helper are required on each shift, the former receiving $4 for eight hours' work and the latter $3·25. The item of fuel and water is wholly relative, but will be given in our particular case, and may be taken as an average for this district.

The following is the tabulated cost for a certain hole which we have taken as an average:—

Shifts of sinking (eight hours each)	23 days	
Depth of hole	308 ft.	
One drill-man, wages		$82·69
One helper and assistance while handling casing		81·08
Three-quarters of a cord of wood each shift, at $4·50 per cord...		77·51
12 bbl. water at $6·50 for hauling, 8 days		52·00
12 bbl. per day (when running two rigs), 15 days		48·75
Coal and oil		7·60
Miscellaneous charges ...		12·20
Superintendence		50·00
Cost of 308 ft. hole		$411·83
Cost per foot.		1·33

Some water was encountered in this hole, and I may say, roughly, that a two-compartment shaft for the same depth, with equipment, would have cost about $12,000, or $40 per foot.

The question has often been raised concerning the character and accuracy of the sample obtained from this work. With small rich veins there may be an objection to the use of drills, but in ore bodies like those found at Ely I believe as good a sample can only be taken with great care. Most of the sludge or drillings will pass a 20 mesh screen, and a good method is to provide a large box with a capacity equal to several screw-lengths. I might say that a screw is 3 ft. long, and when fed out the clamps are changed on the rope, and the hole baled or sand-pumped. By settling and decanting the water the entire product of the hole may be saved and sampled. In ordinary practice it is sufficient to dip a sample from the box and pour the same into a box partitioned off in compartments about the size of common brick. This sample, when dried in the sun, is compact enough to be sampled by chipping, and can be shipped and carried about without breaking.

In a hole carried down without casing there is danger of knocking down particles from the upper portions of the hole. This will vitiate the sample, but the harm done is more theoretical than actual. As is well known, the ore bodies of Ely are the impregnation of a great stockwork of porphyry, and an ore body of much value must be a hundred or more feet thick. While actually in ore the sampling must be done with extreme care, but often the hole, all or in part, may be in barren country rock, and only a knowledge of the formation penetrated is desired.

The best test where not in ore is obtained by panning the sample. This concentrates the coarse particles, which are clear to the eye if a magnifying-glass of low power is used. In oxidised formations some doubt often arises as to whether the drillings are composed of porphyry or limestone. A small bottle of dilute hydrochloric acid will soon settle this question with entire satisfaction. We have made a practice of saving a sample of each formation penetrated in 4 oz. bottles. This gives a clear picture of the hole, and is valuable for future reference.

Our drills are provided with traction gears, and may be moved at will over the roughest ground. This saves the expense and trouble of procuring teams. The winter weather is severe for outside work: and sectional houses, so made as to be readily knocked down and moved, have been constructed to enclose the drills, and within these houses the men work in comfort.

The cost of a No. 3 drill is as follows:—

Drill f.o.b. Beaver Falls	$1,400
Cost of casing for 400 ft. and fishing tools ...	350
Freight to Cherry Creek, Nevada	328
Hauling sixty miles from Cherry Creek ...	225
Cost of No. 3 drill, good for 500 feet, laid down on the ground	$2,303

A drill of this size is light for holes of greater depth than 500 feet. The size recommended is a No. 5, good for 1,200 ft., and the cost of the same with traction gear is about $2,500 laid down at Ely. Should five holes of 500 ft. each be sunk, the cost of equipment per hole will be $500, about the price of a good whim outfit. The cost of $1·50 per foot is so much cheaper than any shaft-work as not be comparable. Water does not retard the work in drilling—in fact, it is a benefit.

I can heartily recommend the use of these drills for prospecting in this region. Apart from the cost per foot, the time in which a good working knowledge of the ground can be obtained is a great factor. The sample is quite as good as a core, and most ground can be penetrated at 25% of the cost incurred with a diamond drill, while the expense of equipment will not exceed 30 per cent. that of the diamond-drilling outfit."—F. S. PHEBY, *Mining and Scientific Press.*—*New Zealand Mines Record*, Aug. 16, 1907, p. 44. (A. R.)

VENTILATION OF WORKSHOPS.—The air supply must therefore be regulated with the help of thermometers. Air in which the reading by the wet bulb thermometer exceeds about 70° begins to cause serious inconvenience with ordinary clothing, and this limit ought not to be exceeded in factories or workshops except under exceptional conditions. Experiments show that if the wet bulb reading rises beyond about 88° in fairly still air, the body temperature can no longer be prevented from rising seriously even in persons stripped to the waist and doing no work: and with muscular work under the same conditions the body temperature may rise rapidly at a wet bulb temperature of 80°. With ordinary clothing this effect is considerably greater. At the upper limits it is not the temperature of the air, but that of the wet bulb thermometer that matters: and provided that the air is so dry that the wet bulb temperature does not exceed the limits specified, air temperatures up to 130° or more can be tolerated without rise of body temperature. Much higher wet bulb temperatures can, of course, be borne for short periods, but the body temperature soon rises seriously. The following observations, made recently by Dr. Haldane, will illustrate this statement: With the air temperature at 131° and the wet bulb at 88° the body temperature remained the same after 2¼ hours. With the air temperature at 89° and the wet bulb at 89° on the other hand, the body temperature rose nearly 3° in the same time, and with the air temperature at 94° and the wet bulb at 94° the body rose 4° in two hours. The subject, who was the same in each of the experiments, was stripped to the waist, and resting. With moderately hard work and a wet bulb temperature of 87°, the temperature rose 4° in one hour.—BRITISH HOME DEPARTMENT REPORT.—*Technical Literature*, Oct., 1907, p. 309. (A. McA. J.)

Petrol-Operated Rock Drill. One of the latest applications of the explosive power of petrol is to the boring of holes in rocks for the insertion of the blasting cartridges. The machine is ingeniously designed, and comprises a small motor in which the vapourised charge of petrol and air is fixed by electricity. The motion thus imparted to the motor is then transmitted to a rapidly reciprocating spindle, to the end of which is fixed the steel drill. At the same time, in addition to its reciprocatory motion, the spindle is automatically rotated, so that the drill strikes a fresh surface at each blow. This tool is unique of its kind, and, being so compact and self-contained, is inexpensive to operate, extremely portable, and easily moved from one point to another, as desired. For quarry work the appliance should be decidedly advantageous, especially as a further improved development upon the tool has recently been successfully accomplished, whereby a battery of drills may be operated simultaneously from a single motor, so that a large surface of rock can be bored at one time.—*Chambers' Journal*, Nov., 1907, p. 767. (A. L. E.)

Electrical Haulage in Mines. "Sir C. Hertslet, in a report to the British foreign office on the coal mining industry of Belgium, writes that some of the mines of the country have adopted electricity as the sole motive force, for extraction, pumping and ventilation, and what are regarded as satisfactory and economical results have been obtained. The question of electric versus steam engines is one which has been much discussed in technical circles, and although it is likely to remain in an indefinite state for some time to come, there can be no doubt as to the superiority of the electrically-driven machine where local conditions are favourable. It is of interest to follow Sir C. Hertslet's remarks, which, on the whole, are favourable to electric machinery, and bring to light the greater simplicity of its manipulation. He cites the following instance:—'One of the events most liable to cause accidents in mines is the possibility of the machinist failing to stop the cage at the exact moment. This is met in a certain measure by the provisions which compel mine owners to have a second machinist when shifts of workmen are going up or down the shaft, but a momentary aberration or indisposition of the machinist at a critical moment might be the cause of a terrible accident before the second man could stop the machinery. In working steam engines the attention of the operator is divided between several control levers, whereas one of the greatest advantages of the electrical machine in this connection is that only one main controlling lever is required. In addition, on the one electric machine which I have seen at work in Belgium, a safety contrivance was attached which, in the event of the machinist omitting to stop the cage on its upward or downward course, automatically brings the starting lever to the zero position and stops the cage.' In the report there is given a copy of the most recent law respecting the lighting of mines, and of the lamps authorised to be used for the purpose. Among the latter may be mentioned the Wolf lamp, used in the mines of the Mariemont & Bascoup Company, which is one of the largest and best-administered mining companies in Belgium. This is a benzine lamp, fitted with an automatic relighter, and, in addition, possesses the advantage of automatically locking after it has been filled and screwed together, so that it is impossible for the workers in the mines to open it, as this can only be done by means of an electromagnet specially used for this purpose. Belgian experience shows no insuperable difficulty in the use of electricity in mining, the advantages of which need only to be persistently urged to receive the recognition of mine managers. With the attention that is now being given by electrical engineers to perfecting machinery, the future should see an undeniable extension of the application of electricity to mining in this country as well as in others."—*Electrical Review* appearing in *Mining Reporter*, Oct. 24, 1907, p. 376. (J. Y.)

MISCELLANEOUS.

Coking of Coals.—"The reasons for some coals coking or submitting themselves naturally and readily to the coking process and furnishing a high-grade coke, are as follows:—(1) They are in structure largely prismoidal, permitting the free mixing of the particles, diffusion of gases, etc. (2) Their chemical composition is such that they contain not only the fusing and cementing factors in the most favourable proportions, but they carry the most desirable elements in those proportions and the form best adapted to the purpose for which the coke is required.

The reasons some coals are not naturally fitted for the coking process but yield high-grade coke after disintegration are as follows:—(1) Their structure is largely of a laminated character, thus presenting obstacles to the free mixing of the particles, diffusion of gases, etc. (2) Their chemical composition is not widely different from those of the natural coking coals, but they cannot adjust themselves to the necessary processes of fusion, cementation, and interchange of position of the different elements the complete coking process requires unless the coal has first been disintegrated, etc.

There are coals that possess all the necessary factors to produce high grade coke, and yet they fail to respond to ordinary treatment, and that is by reason of two different grades of coal being embraced within one seam—one being extremely rich, and the other being lean—with a consequent difference of densities which has the effect of separating the respective qualities when the coal is put into the coke ovens in the ordinary way. The writer had an experience with a coal of this kind, the chemical composition of which was extremely good, but physically the resulting coke was weak and, as a consequence, unfitted for a smelting fuel. A system of disintegrating and pulverising was installed with the result that the product was of A-1 character, both chemically and physically. This is another illustration of physical effect carried by the coal and inherited by the coke."—F. C. Keighley, *Mines and Minerals*—Coal Mining Institute of America, Oct., 1907, p. 109. (A. McA. J.)

Carrying Capacity of Launders.—"The author writing in the *Mining and Scientific Press* says: I can positively state that the 'wetted perimeter' has nothing whatsoever to do with the carrying capacity (as far as solids go) of a launder. Personally, I was formerly under the impression that it had, but my experiments have conclusively proved that we have all been wrong in this assumption. For example, a 1 in. launder using 25 lb. water per min. will carry exactly one-half of the tailing of a 2 in. launder using 50 lb. water and one-tenth of a 10 in. launder using 250 lb. water per minute, all, of course, on the same grade: that is, the carrying capacity in sand, etc., is proportional to the width. This I can prove by figures from about 500 actual experiments."—G. A. Oserstrom.—*Mining and Scientific Press*, Sept. 14, 1907, p. 332. (K. L. G.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

ELECTRIC BLASTING APPARATUS AND EXPLOSIVES. By WM. MAURICE. Price, 8s. 6d. (London: *The Electrician* Printing and Publishing Company, Limited.

"This book deals with its subject a practical detail, clearness, and conciseness which is beyond all praise. It is designed more especially to give those who have to be responsible for shot firing by electricity in collieries and understanding knowledge of their art.

Illustrations and diagrams of practically all the different exploders in general use are given and clearly explained, therefore there will be no excuse in future for the facts stated by the author, if fact it be that the average shot firer has no idea of how his machine works. The chapters on care and manipulation of cables, testing fuses and other apparatus, are valuable, and the theory of the cause of gas and dust explosions in mines is described in an interesting way. The list given of the composition of the various 'permitted' explosives, and the information on the Acts of Parliament which regulate the use of explosives in mines, render the book complete and useful for reference.

The author is to be congratulated on producing a really valuable book about a subject upon which there is not much literature, and what there is, is so scattered as to be difficult to refer to." –*London Mining Journal*, Nov. 30, 1907, p. 671. (A. R.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 31/08. Frederick Charles Hendricks. Randite. 30.1.08.

(P.) 35/08. Andrew Frederick Crosse. Improvements in the extraction of gold and silver from slimes and in means therefore. 1.2.08.

(P.) 36/08. Andrew Frederick Crosse. Improvements in the treatment of cupriferous ores containing gold and silver. 1.2.08.

(C.) 37/08. Otto Tonnesen. Improved bolt for survey peg and other purposes. 3.3.08.

(P.) 38/08. Thomas Speedy. Improvements in fixing tappets to the stem of gravity stamps. 3.2.08.

(P.) 39/08. James Malcolm. An improved rod for tamping explosives. 3.2.08.

(P.) 40/08. Alfred Edwin Davis. Improvements in and relating to rock drilling machines. 4.2.08.

(C.) 42/08. Mortimer Frederick Mieville. Improvements in apparatus for the production of carburetted air. 7.2.08.

(C.) 43/08. Ernest John Sweetland. Improvements in filters. 7.2.08.

(C.) 44/08. Albert Edward Hodder. Improvements in or relating to automatic water elevators. 7.2.08.

(P.) 45/08. Sydney Manthrop Cockburn (1), John Ridley Temperley (2), Joseph Temperley (3). Improvements in water-tube steam boilers. 7.2.08.

(P.) 47/08. Henry Howard Johnson. Improvements relating to cam shafts for gravity stamp batteries. 7.2.08.

(P.) 48/08. James West (1), Robert Wilkinson Walton (2). Improvements in grips for cable haulages. 10.2.08.

(P.) 49/08. James Richard Whiting. Improvements in and relating to rock and like drilling apparatus. 10.2.08.

(P.) 50/08. Robert Thomas Whitfield. Improvements in means for hanging stamps. 12.2.08.

(C.) 51/08. Thomas Mathieson Thom. Improvements relating to the manufacture of stone or marble artificially. 14.2.08.

(C.) 53/08. David Burnett. Water valves. 15.2.08.

(P.) 55/08. Hebert Hulford (1), Melvin Frazer (2). Improved petrol economiser and carburetter control. 15.2.08.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

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JOHNSON, TOM, *l/o* Denver; c/o Mr. Jack Johnson, Grammar School, Wen, Salop, England.
MAURICE, R. F. G., *l/o* Fordsburg; Sabi Mine, Belingwe, Rhodesia.
METCALF, J. E., *to* P. O. Box 108, Germiston.
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Vol. VIII. MARCH, 1908. No. 9.

Proceedings
AT
Ordinary General Meeting, March 21, 1908.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, March 21st, Prof. J. Yates (President), in the chair. There were also present :—

35 Members : Dr. J. Moir, Messrs. T. Lane Carter, R. G. Bevington, W. R. Dowling, K. L. Graham, A. Richardson, G. O. Smart, H A. White, Prof. J. A. Wilkinson, A. A. Coaton, M. H. Coombe, J. Gray, A. G. Hoyer, H. Johnson, J. A. Jones, J. Kennedy, M. Knight, G. A. Lawson, J. Lea, J. P. McKeown, J. E. Metcalf, J. T. Milligan, T. T. Nichol, F. B. Ogle, J. F. Pyles, S. C. Quin, W. H Roe, C. B. Saner, M. Smith, W. H. Smith, S. H. Steels, H. Taylor, J. A. Taylor and J. F. Walker.

10 Associates : Messrs. J. Chilton, C. W. Dowsett, W. N. Dunnachie, A. L. Edwards, J. H. Hawthorne, R. W. Leng, A. Thomas, W. Waters, E. M. Weston and L. J. Wilmoth.

7 Visitors (including Mr. H. Kestner) and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting as published in the *Journal*, were confirmed.

NEW MEMBERS.

Messrs. Whitehouse and Smith were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that the candidates for membership had been duly elected, as follows :—

Boright, Sherman H., Great Eastern Collieries, Ltd., P. O. Box 5, Springs. Mining Engineer.
Cellier, Jacobus Stephanus, M.E., Consolidated Gold Fields of S.A., Ltd., P. O. Box 1167, Johannesburg. Mining Engineer and Government Land Surveyor.
Glyn, C., Glen Deep. Ltd., P. O. Box 184, Germiston. Mine Manager.
Inglis, Malcolm Canmore, New Club, P. O. Box 1078, Johannesburg. Manager, Molyneux Mine.
Klein, Dr. Max Marcus, Brakpan. Medical Practitioner.
Lightbody, Thomas, Luipaardsvlei Estate & G. M. Co , Ltd., P. O. Box 268, Krugersdorp. Tube-miller.
Mitchell, Andrew, East Rand Proprietary Mines, Ltd., P. O. Box 80, East Rand. Battery Foreman
Neale, Arthur Leggett, Transvaal G. M. Estates Ltd., Pilgrim's Rest. Mining Engineer.
Pearson, Thomas, Driefontein Deep, Ltd., P. O. Box 23, Knights. Mine Manager.
Williamson, Robert, Luipaardsvlei Estate & G. M. Co., Ltd., 21, Luipaard Street, Krugersdorp. Tube-miller.

The Secretary announced that the following gentlemen had been admitted as Associates by the Council since the last general meeting :

Anderson, James Stuart, Dharwa Reefs Co., Ltd., Kabulayatkatti, Dhwarwa District, Bombay Presidency, India. Reduction Officer and Cyanide Chemist.
Dowsett, Charles William, Witwatersrand Deep, Ltd., P. O. Box 5, Knights. Cyanide Foreman.
Freusberg, Adolph, Lebong Tandai (Ketoun), West Coast, Sumatra. Assayer.
Herald, Andrew James, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Mill Foreman.
Linton, Robert, Banco Del Oro Mining Co., Magdalena, Sonora, Mexico.
McCrindle, James, Knights Deep, Ltd., P. O. Box 143, Germiston. Amalgamator.
Murphy, Thomas Daniel, Ventanas Mining and Exploration Co., Ltd., Ventanas (via Chavarria), Estado de Durango, Mexico. Mining Engineer.
Pryce, Wynn St. Lawrance, Treasury G. M. Co., Ltd., Cleveland. Assayer
Thayer, Harry Stanley, U.S. Reduction and Refining Co., Ltd., Colorado Springs, Colorado, U.S.A. Assistant Superintendent.
Thomas, David John, Simmer and Jack Proprietary Mines Ltd., P. O. Box 192, Germiston. Amalgamator.

And as Students :

Ball, Harry Standish, P. O. Box 2536, Johannesburg.
Davies, Cyril Nutcombe, Halcombe, Jubilee Road, Parktown, Johannesburg.
Donovan, Donough Bennitt, 103, De Korte St., Wanderers View, Johannesburg.
Gilbert, Thomas Woodford, 47, Bok St., Joubert Park, Johannesburg.
Graham, Harry Woodthorpe, P. O. Box 1155, Johannesburg.

Malherbe, David Gabriel, c/o Mr. J. H. Koch, 40, Buxton St., Doornfontein, Johannesburg.
Spargo, Richard James, Transvaal University College, P. O. Box 1176, Johannesburg.
Sutton, Joseph Bennet, 10, Highfield Terrace, Doornfontein, Johannesburg.
Tapscott, Sidney, 10, Highfield Terrace, Doornfontein, Johannesburg:
all are students of the Transvaal University College.

General Business.

The President: Since the last meeting we have had a really very interesting trip to the Apex Colliery, and not the least enjoyable part of that trip was the delightful little speech delivered by Mr. Dalrymple. I move a vote of thanks to the Company for their hospitality. It is perhaps as well that I should call your attention to the special meeting we are to have on April 18th. The purpose of that meeting is specified in the agenda. Another item on the agenda is the contemplated visit to the Nigel. We have had very many invitations to mines and works in the past, but I think I am quite right in saying that we have never had an invitation which exceeded in cordiality this one, and we hope it will be a great success. The idea is to have a special train, and that, of course, will necessitate a certain number coming forward—65 in this case.

LETTER.

A letter was read from Mr. W. Sinclair Brown, Lebong Soelit, Sumatra, taking exception, as the metallurgist concerned, to the remarks made by Mr. A. F. Crosse, published in the September *Journal*,* in which it was stated that nitric acid was used to dissolve the copper out of the base bullion, and that the silver was also dissolved which accompanied the copper "down the creek." Mr. Brown pointed out that the copper and the silver were duly recovered and accounted for, as could be ascertained by reference to the company's books, and asked that Mr. Crosse would either withdraw or amend his remarks.

The President: I think that when these facts are placed before Mr. Crosse, he will gladly make his reply.

ASSOCIATES' SUBSCRIPTIONS.

The Secretary: With reference to the proposed raising of the Associates' subscription, the reason the Council has had this under consideration is that at present the Associates are not paying their full share of the expenses of the Society. The cost, roughly, per member, is in the neighbourhood of 27s., and thus the fact that the Associates only pay 21s. makes a considerable difference to the Society's finances. That is why the Council has considered the matter very carefully. They now make a proposition to raise the subscription to £1 11s. 6d. If the motion is successful, and the amount is raised, it will make a considerable difference to the finances, and the difference will be seen, we hope, in the size of the *Journal*.

SAMPLING APPARATUS.

Prof. J. A. Wilkinson: The apparatus which, I am given to understand, has hitherto been used here for sampling the gases occurring in mine air, consists of a Winchester quart bottle and a pair of bellows. The bottle is taken to the place, whence a sample is desired and the air pumped into it by the bellows, after which it is corked and taken to the laboratory for analysis. It is obvious that in this case before any gas can be removed from the bottle, it must be uncorked, and it occurred to me that it might convenience the analyst, whilst at the same time allowing a representative sample to be taken if this could be avoided. The apparatus proposed consists of a strong Winchester quart fitted with a tight india-rubber stopper bored with two holes, through one of which a tube passes to the bottom, and through the other a tube passes just below the cork. On the outside, these tubes end in good glass stopcocks with a short prolongation of the tube beyond the cock. The cork fitting into the bottle is wired down to the neck and then carefully sealing-waxed, so as to be quite certain that no leakage will occur. The taps of the glass stopcocks are also wired. Before using, the Winchester must be thoroughly cleaned and dried. Then it is evacuated by means of a good pump, as far as possible. I have found in practice that a good water pump is sufficient for the purpose, a high vacuum being unnecessary. In sampling, the tap attached to the tube which passes to the bottom is opened and the gas rushes in. The other tap is then turned, and to the other tube is attached a small india-rubber pump, such as is used for spray bottles. This is worked for a short time and then the taps turned off.

Dr. J. Moir: This seems to be a good method, but I should like to ask whether Prof. Wilkinson has tried whether such bottles will stand the vacuum. Secondly, what would be the best method of expelling the gases?

Prof. J. A. Wilkinson: It is impossible to say that every Winchester will stand a vacuum, and the only answer possible is the one given by experiment. I used an ordinary German Winchester, and found it to stand satisfactorily the vacuum created by an ordinary glass water pump. This, however, was not very great, and hence I was compelled to use the small spray pump. To expel the gases for use in the laboratory all that is necessary is merely to run in some displacing liquid through the tube which passes to the bottom of the bottle, either water or brine or glycerine, etc., according to the analyst's choice.

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* Vol. viii., p. 80.

NOTES ON RAND MINING.

By Tom Johnson (Member).

As you are aware, great exertions are being made at the present time to reduce the working costs on our mines, and it has very often occurred to me that something could be done towards this very desirable end by taking lessons from other places and other industries.

I think it is generally believed, and rightly, that where the greatest saving is to be looked for is in departments of the mine other than the reduction works; not that a saving cannot be effected in the reduction works, but that a greater saving can be effected in the other departments. For one thing, it has always been my opinion that there are too many shafts, winding engines, etc., for the amount of rock to be hoisted. Many hoisting engines do little beyond condensing steam, and I am glad to see that, latterly, there has been an improvement as regards the number of shafts, but there are still too many hoisting engines being provided.

It is interesting to consider what is the real determining factor for the size of shafts; and I believe I am right in saying that the general idea is that the amount of rock to be hoisted is the factor; if so, it should not be: the determining factor should be the size necessary for ensuring good ventilation, because if a shaft is large enough to give good ventilation it is large enough for hoisting the rock through. It should not be impossible to hoist 5,000 tons of rock per 24 hours through two compartments, 6 ft. × 5 ft. from a depth of 3,000 ft.

Now a shaft with three compartments of that size would be large enough for hoisting that amount of rock and still leave room for a pump compartment. It would require a ventilating pressure of 25 lb. per square foot of area, (taking the coefficient of friction at ·005, which is very low for a timbered shaft) to pass sufficient air for the number of persons required to handle this amount of rock; this is for this shaft alone without the workings or other shaft; so it can be seen that although the rock could be hoisted through a shaft of this size the air could not pass through except with an extravagant expenditure of power. Some may doubt the possibility of hoisting this amount of rock through two compartments, but I would point out that at Kimberley it is stated that 2,300 tons were hoisted from a depth of 1,200 ft. in 8 hours; if this can be done at Kimberley I cannot see any reason why as good work cannot be done here.

At the Baggeride Colliery, South Staffordshire, the 8 hours' output is expected to be 3,584 tons; the depth is 1,900 ft., load 7·28 tons, total load 19 tons, maximum velocity 3,900 ft. per minute, acceleration and retardation periods being 7 seconds each. The Delaware and Lackawana and Western Railway Co. have a water hoist which runs automatically; the total load is 27 tons, weight of water each trip 33,180 lb., depth 550 ft., time of hoisting one minute, total time per trip 1½ minutes, giving an output of over 11 tons of water per minute; I believe this hoist works more cheaply than pumps.

I give these examples just to show how far we have to travel yet before we can take our place in the front rank as mining people. Some may think, what is the use of talking of hoisting 5,000 tons of rock per 24 hours through two compartments if we shall never want to do it; but I think we shall as the 1,000 stamp mills that are spoken of will need a daily hoisting of, say, 12,000 tons of rock; in any case it is a useful lesson to know what is actually being done by others, and also to compare our efforts with theirs. When we look around and see two, three and sometimes more hoisting engines, with the necessary enginemen, Kafirs, wear and tear, in use for the hoisting of, say, 1,500 to 1,600 tons of rock per 24 hours, it is time to pause and ask ourselves if we are doing as good work as we should or could.

There are a number of shafts where a balance rope under the skips or cages would save a little, as balancing the load on hoisting engines tends to quicker, safer, and cheaper work. All hoisting engines should be fitted up in such a manner that the engineman can do his work either sitting or standing, and should have looking-glasses fixed so that the engineman can see all parts of his engine from the platform; it is just as well for work to be easy as hard. They should also be fitted with some controlling apparatus, such as the Karlik-Witte or Nicholson's, both American inventions, the Visor, Lancashire, or one invented by a French engineer, which is in use on the Continent. Any of these will prevent an overwind, save the expense of accidents and allow faster hoisting, as no matter how fast the engines may be running, when a certain point is reached, if the engineman does not take charge, the controlling device will. Speaking of fast hoisting, Mr. Stokes, an inspector of mines in England, I believe, sometime ago wished to find out how fast men were being hoisted at some of the first class collieries in England; from his experiments and observations he found that in some places the speed was as high as 37 miles per hour, say, over 3,200 ft. per minute, in the middle of the wind.

All shafts should be fitted up with good telephones in addition to the electric signalling apparatus and from the same points; I have seen cases where many pounds could have been saved if the shafts had been fitted up with telephones, and they would always be useful and time saving

As to shafts, are we right in keeping to the rectangular form: if so, are we right in putting in so much timber, a great amount of which is of little use? I think vertical circular shafts are much stronger than rectangular ones and would stand better without lining after reaching the hard rock, except where faults, etc., are met with: all dividers, etc., could be kept out and wire rope guides used which should hang clear from top to bottom. Rectangular shafts when sunk across the strike should not need timbering all through: wire rope guides could be used in these shafts also. If shafts were too small to give the necessary room or clearance for the swing of rope guides, then rail guides could be used, two for each cage, the bearers to carry the guides being on the outer side of each cage or skip. It will doubtless be said that guides put in in this manner would prevent the use of a bend; I should say a good thing, too, as I have always believed it a fallacy to think that bends save money. I think it better policy to sink further if necessary and crosscut to the point where the incline shaft should start. Bends are bad things anyway, and there have been many accidents that would not have given half the trouble if a bend had not been in the way. There was one recently which, if it had not been for the bend, would not have stopped the shaft above 5 hours instead of as many days. Bends wear out ropes, take a great deal of attention and are bad to get men through. In my opinion they waste money instead of saving it.

On inclines of a dip of 45° and under, I advocate endless ropes and ordinary cars instead of skips, as in this way the vertical shaft load can be conveniently balanced, and the incline load, being also balanced, more rock can be hoisted through the same sized shafts: this is in place of compound shafts. Using endless rope and cars on the inclines, a smaller shaft is permissible for a maximum tonnage, weight and cost of rails, sills, etc., is less, speed lower, and output more regular than with skips.

If the incline were driven in the footwall, as it should be, the cost per foot would be less, and the amount of waste rock to be hoisted would be less for an endless rope-way than for skip-ways: 1,000 tons could be hauled up an incline 3,000 ft. long in 12 hours by endless rope and cars on a pair of tracks requiring a width of 7 ft.; with pump and travelling way the width would need to be, say, 13 ft. over all, and with two pairs of tracks and pump-way it would need to be 22 ft. The engines and boilers could be placed underground, if necessary, or the engines alone, power being conveyed from the surface. If boilers were used underground, the waste heat would provide ventilating power, this is done in quite a number of mines in England, and in some places men are regularly hoisted in the upcast shaft. Boilers underground would, I think, be the cheapest method of producing the power for the hauling engines, pumps, fans, etc.

If on any property two shafts had to be sunk they should be put down as close together as the law will allow, a connection being made between them every few hundred feet for ventilation; if water was met with the temporary pumps could be placed in these connections: afterwards permanent pumps would be necessary in one shaft only, but could be reached from both shafts: this would be much easier and cheaper than putting pumps in both shafts, as has been required at many mines.

Having the two shafts close together and the drives rising outwards from one point, all the water above the head of the inclines would drain to this point, and the whole of the rock would also come to it. If an accident happened at one shaft the other shaft would be available for the whole of the rock. At present we generally have a rise of a few feet between the two shafts making it very hard to get all the rock to one shaft if anything happens to the other. In this connection, it is very noticeable how very rarely accidents happen to the shafts, ropes and engines, in the colliery districts in England. Another advantage of sinking the two shafts close together is that, being connected quickly, the development could be done cheaper as the mine would be cooler and more comfortable to work in. It might be said that development would be restricted, as there would only be practically one shaft to commence from, but I do not think this would be much felt in practice, as during the time the shaft boxes and stations were being cut out, the main crosscuts could be driven to where the head of both inclines, if two were necessary, would be, and the two inclines would soon diverge, thus providing room for development.

If it is possible to make arrangements with some other neighbouring company to provide an outlet, only one shaft should be sunk. In 1906 this arrangement was spoken of as a good one for the Simmer West, but was thought to be too far-fetched an idea then, but now I fancy opinions are altering, as it is getting to be the recognised thing. The possibilities of endless rope work in inclines for working the lower deeps do not seem to be generally recognised, but I believe any depth possible to be worked can be served by inclines and at a lower cost per ton than by vertical shafts. The idea would be to start from the outcrop and cut the shafts in the floor of the stopes of the bottom reef, packing the rock in the stopes, or the shafts could be in the solid underneath the bottom reef. I do not think there would be much danger in trenching the shaft in the floor of the stopes, either way the shafts could be started at different points in their length, and could be finished in the time it takes to sink in the solid from the lowest or nearest point to the new ground to be opened up. If thought better, inclines could be started from the vertical shafts of the lowest deeps and carried down to as great a depth as was found possible to work at, instead of sinking the very deep shafts (6,000 ft.) vertically; I believe it would be better, safer, and less difficult, to have stage winding in the incline than in the vertical.

As an instance of what is meant, take the line of the Simmer Deeps, Simmer West, the Jupiter and the lower side ground; this could all be worked with a couple of long inclines, endless rope, say, in stages of 4,000 ft., the shafts being about 7,000 ft. apart, and the same could be done all along that line.

For endless rope incline work very economical engines could be used, as the work is practically continuous, and steam could be kept easier, as there would not be the sudden pull on the boilers as in vertical hoisting. If one pair of tracks can handle about 1,000 tons in, say, 10 hours, three pairs of tracks could handle 3,000 tons per shift, or between 5,000 and 6,000 tons in 24 hours and handle the labour force as well. The ropes in the incline would need to travel at a speed of about three miles per hour. I have heard it said that these large quantities could not be got to the shafts, but I think, if the will to do it is there, the means can be found: at present on many levels of the mines not one-sixth of the possible amount of work is done. A 200 stamp mill should be kept going off three levels at the most, as when too many levels are open the consequent dispersion of labour and waste in supervision is a serious bar to economy.

Stoping.—In stoping, I believe, it would conduce to cheaper work if more men and machines were employed on each stope face. I have many times seen two machines on a 200 to 250 ft. length of face, probably breaking about 16 tons of rock per shift, where there was room enough for machines to be breaking up to 100 tons per shift in many cases. In most mines there are too many stope faces open, causing the work to be scattered and difficult to look after, whereas if the work were more concentrated it could be much better looked after by all concerned, from miner to manager; moreover the larger tonnages coming from each working place would make it advisable to take more advantage of mechanical means of handling rock, such as brake-roads, aerial gears, shaking shoots, mono-rails, etc.; the drives could be made wider, and either air, oil, electric motors or other mechanical means of haulage used for transporting the cars to the shaft and back, instead of the black human motor: good handy boxes and good tracks would have to be put in, so that large quantities could be handled comfortably.

If this system of putting large numbers of machines, or other means of breaking large quantities per shift on each face, were to become general, the drives could be further apart as also the winzes, because the blocks would be worked out quicker, and therefore could be formed larger. The policy to adopt in opening out would be to set a certain area of ground apart to start the mill from, push out the drives to the boundary, then commence stoping there and work home, taking out everything, and leaving no pillars in the stopes, which could then be left to cave, or be packed with waste or tailings, ashes, etc.

Each stope face should be in charge of one man who would be responsible for the direction of the face, of the holes, and the burden put on them, also for the amount of explosives used: he having other men under him to do the drilling, take down the machines, and clear the gear, blast, make the place safe, and rig up again for the next shift of drillers. These drillers would work, say, an eight hours' shift at drilling only, this would necessitate what could be styled the "rigging gang," working in between shifts.

To enable large numbers of machines to work in the stopes from one drive at one time, large air pipes would have to be installed, and to save loss of air good fittings should be used, bends instead of elbows, Tees y'ed on the branch outlet and made longer, and valves of full area right through. Globe valves are bad, for they cause changes of direction; plug cocks give a straight run, but most makers spoil them by reducing the area through the body, the hole through the plug is fair in many, but the ingress to and the egress from the plug is very contracted. A short piece of rubber hose makes a good flexible joint where pipes are led into a stope, raise or winze: the ordinary 1 in. unions that are generally used for connecting the inch pipes and hose have not enough thread on them, and the thread soon strips off and the union has to be thrown away after much air has been lost by leakage; it is much cheaper to buy higher priced unions with a better and longer thread.

As regards breaking rock, it is good policy to break until the stope is full of rock, as it is called; but should the rock get in the way of the machinemen, some or all of them could be put into another place for a time. By having plenty of rock in a stope, a barrier is formed between the flying rock of the blasts and any appliances put in to facilitate the removal of the rock; this protection makes it unnecessary to take them down after every shift, thus saving time and increasing their usefulness, while another advantage is that each shoveller can move more rock per unit of time when there is abundance of it. No pillars need be left in the stopes, only ribs by the sides of the drives for a time or stulls, until they are packed with waste or tailings.

It has been freely stated lately, that packing stopes prevents subsidence, but this statement is rather ambiguous; packing certainly reduces the amount of subsidence but does not prevent it; with closely packed places, the material being shovelled in, the amount of subsidence is about 50% of the height of the excavated area, and will probably, in the greater number of cases, travel to the surface with little diminution. From the caving that has already taken place on the Rand we cannot tell how the surface will be affected in the future. It is thought in some quarters, that when an area caves the weight is gone, and the other parts are safe; it is so for a time, but it must be understood, that when a series of pillars are crushed out and the area caves, the remaining pillars have an increased weight thrown upon them, since they have to support the area above themselves and also form piers carrying the bridge formed across the caved area; this happens in all cases where the caving does not follow fault planes to the surface on the first caving. One of the greatest dangers to a mine is where a caving has taken place right up to the pillars of a shaft, as these pillars will then have to take the extra weight and will start flaking and crushing, and being, as a rule, fairly large, the weighting will not be noticeable for some time. To help these pillars, packs should be built alongside them on the side nearest the caving, and packs should also be built on top of the fallen rock up to the new roof, so as to help to take the weight when the new caving commences. In this way the shaft can be relieved somewhat by throwing the weight into the caved area again.

In a few years' time we shall have some trouble from caving, for at present it has been local and has been little to what we may expect later, as the more area excavated the less there is left to support the cover. To minimise this trouble packing should be commenced now; the pillars recovered would probably pay for the packing, and tailings would be the best material, as they would lie very close, and would be ideal material for flushing into a caved area to pack it up: it is important that the outcrops should be packed not only to prevent caving but to insure the impounding of water after abandonment.

Reverting again to the breaking part of stoping, it might be said that by handling large quantities of rock from each face the grade could not be regulated so well as at present, but it should be quite as easy, for if you take a certain amount of rock of nearly the same value from several stopes, it is the same as if all the rock of a similar value were taken from one stope only; in any case, we could readjust machines, etc., as at present. Speaking of grade, naturally brings to mind the thought of how best to improve it; now there are two good ways of improving the grade; first, break the reef as clean as possible; second, if waste is broken, sort out as much of it as possible, either underground or on the surface: the point which concerns us most is which is the cheaper to do, not to break waste, or to break a certain amount and sort it out after. To my mind, the cost per ton bugbear has been responsible for the loss of a great deal of money, for low costs are not always a measure of good work, and in many instances it would pay to let the costs come up if it were only done in a proper manner.

Case I.—For example, say, there was a mine where the recovery value of the rock was 20s. per ton and the costs 20s. per ton, stoping width 5 ft. capacity 30,000 tons, and everything sent to the mill: the breaking cost of this rock would be, say, 5s. per ton or, say, 76s. per fathom, the profit being, of course, nil.

Case II., same Stopes.—Now if it was found possible to stope all the reef out with a stope width of 4 ft, what would be the value, cost, profit per ton and total profit, on the handling of 30,000 tons per month as before? Say, in this case we allow 3s. per fathom more than in Case I. for breaking, because of the narrower stope width or, say, 79s. per fathom, that is 6s. 7d. per ton broken; instead of 5s. 1d. as in Case I. The value would rise from 20s. to 25s. per ton, the cost from 20s. to 21s. 6d., the profit from nil to 3s. 6d. per ton, and the total profit from nil to £5,250. That is to say, it is possible to increase the profit by increasing the cost if it is done properly. If in these two cases the recovery value at the beginning had been 30s. instead of 20s. the profit in Case I. would have been £15,000 and the increase of profit in Case II. would have been £9,000 instead of £5,250.

Case III.—We will now look at it another way; we could still break a 5 ft. stope, but sort out as much of the extra 1 ft. of waste as possible and stow away in the stope, this would apparently save some of the higher breaking costs; under these conditions what would be the result? The breaking costs would be 5s. 9·3d. per ton sent out, only 60 % of the 1 ft. of waste could be sorted, as the remainder would be too fine; the cost of sorting and stowing would probably be about 5s. per ton handled. As there is only a small amount, the value of the rock sent out would be increased to 22s. 8·64d. per ton, the costs would be 21s. 4·4d. per ton, profit 1s. 4·3d. per ton, and the total profit about £2,025.

Case IV.—In this case we break a 5 ft. stope and sort on surface instead of underground; it must be remembered that I am only dealing with waste outside of the reef, the partings are quite different. As in the last case only 60 % of the 1 ft. of waste could be sorted, hence breaking costs would be the same, 5s. 9·3d. per ton crushed: value of rock to mill would be the same, 22s. 8·64d. costs would be 22s. 2d. per ton, profit 6·6d., total profit £816 or, say, 4s. per ton for the tons sorted, being the difference between cost of sorting, 10d. per ton, and cost of reduction 4s. 10d. per ton. The small amount of saving to be effected by sorting on the surface deserves special consideration, as I believe that many men are not as careful of breaking waste as they should be: they say, "Oh, a little waste does not matter, it will be sorted out on top," forgetting that it costs 13s. to 14s. per ton before there is a chance to sort it on the surface. Given that I have fixed the costs fairly for the different cases it can be seen how important it is not to break waste, and if broken it should, if possible, be packed below. It may be thought that I have gone to extremes in basing the figures on being able to cut out a foot of waste: I have used a foot merely as an example, and those who cannot cut out a foot should cut out what they can, even if it is only 2 or 3 in. Profits are what are wanted, and if better profits are got in this manner there should be no kick about the higher costs

In stoping with two reefs close together, say, 3 to 6 ft. apart, it would probably be best to stope out the middling with machines and stow it, then break the top and bottom with hammer boys, or substitute, if the reefs are thin, or large machines if thick: if preferable, one reef may be stoped, then the middling broken, and lastly the other reef, as has been done at the Geldenhuis Deep for years: or it may be found more profitable to stope out the bottom reef, then pack with tailings flushed in, afterwards working the higher reef.

I do not know what the cost per ton of rock extracted would be to flush this packing material into the mines; on the Continent of Europe it costs less than 6d. per ton, even where they have to get the material and transport it to the top of the shaft. On the Rand, I believe, we could get our tailings to the shaft head much cheaper than to the dump, especially in places where the cyaniding tanks are higher than the head of the shaft, as here the sand could be flushed out of the tanks into launders: in other cases where the shaft head is higher a small tunnel could be driven from the shaft to the tanks, in others again a short borehole from the tank site down into the workings would be best.

The life of the pipes is longest with fine material, and the Pohle air lift for tailings should give a guide as to the length of life of vertical pipes: the lines of pipes on the drives would be the ones to suffer most, but the elbows could be protected more or less by forming ribs on the inside, the sand would partly fill up between the ribs and help to protect the elbow, and short lengths of pipe should be put in on each side of the elbows to facilitate their removal. At the Myslowitz Colliery, Upper Silesia, Germany, they found that by lining the pipes with wood, ¾ in. thick, and turning them round at different times, the wood lining lasted 10 months. The size of pipes used are from 6 in. to 8 in. diameter, the quantity of water to sand 1¼ to 1, the quantity of material flushed being 35,000 cub. ft. per day down one pipe. Here on the Rand the workings would about hold the quantity of sands produced.

In the mines of low dip it would not take much strength of barrier to hold the packing or flushed material: but in mines of high dip a few experiments would have to be carried out to find out if the ordinary stulling would be strong enough; of course, where the drive did not need keeping open stulling and everything of value would be cleared out and the barrier put in the drive.

This system of packing old workings and saving surface area is no new thing and is used in many places, such as the Pas de Calais group of collieries, France, the Deutscher Kaiser mine, Westphalia, where they use granulated blast furnace slag, the Klein Rosseln mine, Saar, the St. Carl and Wendal, and Salmbach mines, also in the anthracite mines, U.S.A.: it was, I believe, first introduced at the Philadelphia and Reading Coal and Iron Company's, Kohinoor Colliery, Shenandoah, in 1887.

Mr. Davies made experiments on the resisting power of culm and found that the settlement due to a pressure equal to 500 ft. of roof was from 2 % to 8 % of the height of excavation, dry culm having the least and wet culm having the greatest settlement. Experiments could be made on tailings in pipes with a hydraulic press or other means of getting pressure. I suggest that the engineering department of the Government Laboratory experiment a little in order to give us some data on this. I guess safety measures have as great an interest for them as for any of us, and they have the means for making experiments.

As to the cost of pumping the water back, such places as the Knights Deep could furnish estimates of cost of handling large quantities. It might be troublesome handling this water where the mine water is not fit to mix with the reduction works water.

As to the manner of carrying stopes, with a dip up to about 40° I like to see the faces of benches bearing away about 30° from the direction of full dip (Fig. I.), greater in flatter stopes. The benches should be broken from the bottom of the stope upwards, instead of downwards as most miners prefer to do: in working upwards the next bench above is lengthened, which is an advantage as it gives room for the explosives to kick beyond the bottom of the holes: in steep stopes the angle of the faces from the direction of full dip would be less (see Fig. II.). Stoping would become more underhand in mines, where it was not advisable to have a large amount of broken ore in the stopes (Fig. II.); but in a mine where the broken ore could be left so as to work on it (Fig. III.), then it would be cheaper to allow the stoping to become more overhand as the dip became greater. As more machines could be got to work comfortably, time would be saved in clearing away the gear, and there would be no danger of the workers slipping down the stope. Also the lower part of the stope being full of broken ore and the workers close up to the face, there would be less chance of accidents. Many accidents happen to boys in the bottom of steep stopes, when underhand stoping, from tools slipping or pieces of rock from the hanging or footwall coming away. Care would be needed to regulate the quantity of rock taken out of the stope so as not to lower it too much, and to see that the rock blasted was broken small enough to pass the boxes freely without choking.

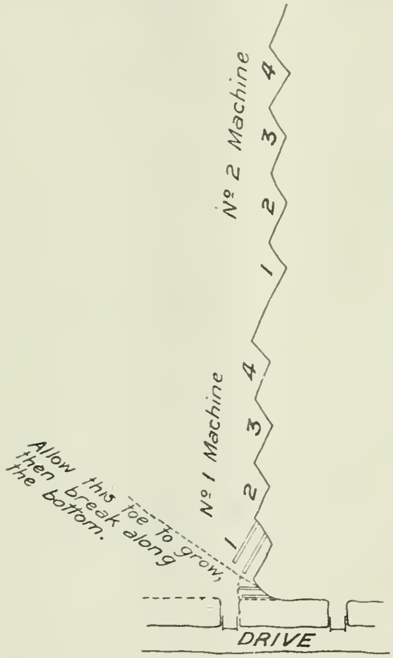


Fig. I.—Part of stope-face showing run of benches 30° from the direction of full dip. Dip 15 to 40. The machines are moved until the four benches are drilled, and then returned to No. 1 bench.

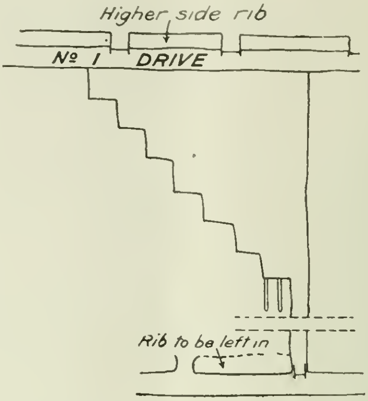


Fig. II.—Part of underhand stope. The top gets too far back to carry light machines, as the broken rock would fall on the lower benches and need shovelling off before drilling could commence.

It might be asked in what manner would it be best to start stoping overhand, whether from the back of drive or leave a rib in? This would depend a great deal on the circumstances of each place; it would certainly be cheaper to start at the corner of the raise and take the back of the drive out and stull. In this case I should take about 4 ft. out along the back of the drive well ahead of the stope, blasting the rock on to a temporary stage just high enough for the cars to go under, putting in the permanent stulling behind, taking care before blasting, either on to the temporary stage or permanent stulling, to get a cushion of rock on (Fig. III.): it would be best to do this breaking with hammer boys or small

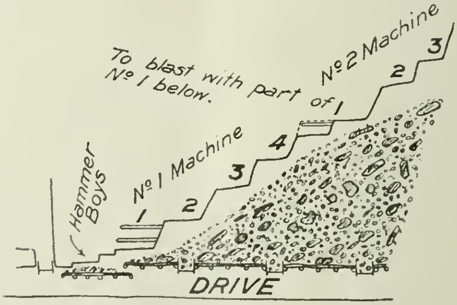


Fig. III.—Part of overhand stope. Benches carried at such an angle that all holes are wet; top or back of drive taken out, and stulling put in. Dip 40 and upwards.

stoping machines, as less rock would find its way on to the drive. If no work was going on beyond the stope I should blast on to the floor of the drive, after covering it with sheets of iron to ease the shovelling.

I know, to many, the idea of using, say, ten large machines on one face is going a little too far, for a 200 ft. back at 30° face angle will give about four benches per machine, and the boss who has men that keep four benches per machine, even if they have 500 ft. of a stope face, should think himself very lucky. It might also be said that drilling so many benches at one time would mean having to blast off full benches, but this is not necessarily so, as in this system special men are supposed to do the blasting and have time at their command.

As to the placing of holes in the benches, there is great diversity of opinion as to which is the best way even amongst good men, so I can only give you an idea of what I have found to be good. In large stopes, over 5 ft. up to 9 ft., I use five holes per bench, two top, two middle and one bottom (Fig. IV.); if there was a floor or parting at the foot or hanging, four holes should do: as to length, the higher the stope the greater the length the holes should be. If there is anything that should be drilled into miners, it is the fact that where the benches are long enough, the putting of burden on the holes by lengthening them is cheaper than the putting of burden on by placing the holes further from the front of the bench. The amount of explosive required is directly as the length of the hole and as the square of the width from the edge of bench to the hole; also the greater the height of the stope, the further can the hole be put from the edge of the bench, that is, in regular ground.

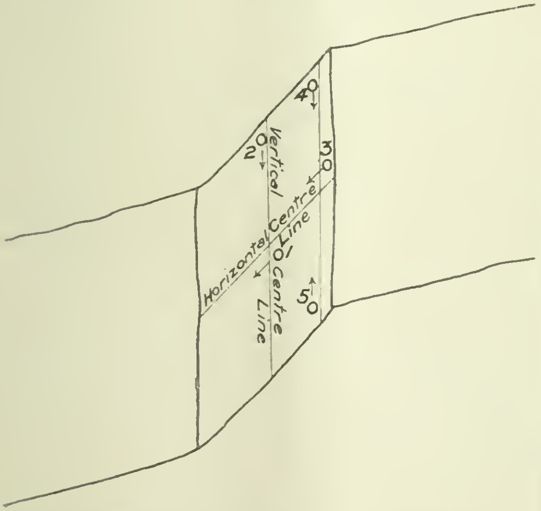


FIG. IV.—Position of holes in stope benches. Stope 8 ft. or 9 ft. high. Bench 6 ft. across. Holes about 9 ft. 33 to 40 tons. Explosives ·5 to ·6 lb. per ton.

In smaller stopes than 5 ft. the holes should be put in two in a row vertically, and staggered in the horizontal line, not the vertical (Fig. V.); I know that this is not usual and is a strange way to many miners, but where there is no parting in foot or hanging I think it the best way of holding the stope narrow and breaking ground. The first bottom hole I should drill some distance from the footwall so that it would have a good chance to break sideways into the stope, the front top hole close to the hanging, and in such a manner that it would break downwards, the back top hole some distance from the hanging so that it would have a good chance to break sideways into the stope, and the back bottom hole close against the footwall and in such a manner that it would break upwards: and I should fire them in the following rotation:—Front bottom, front top, back top, back bottom: this would keep a neat stope and would not damage the hanging. With a parting either in foot or hanging I should drill the breaking out hole next the parting: with a parting in both foot and hanging it could be often brought out with one hole, if decent judgment was used in fixing the burden.

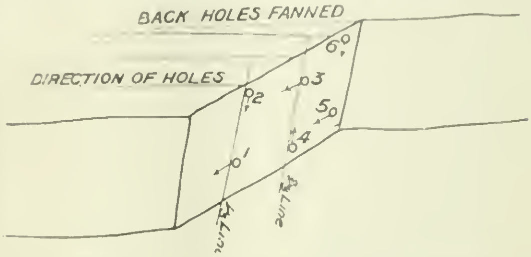


FIG. V.—Position of holes in stope benches. Holes staggered horizontally; no partings. Stope 3 ft. high; bench 6 ft. across; holes 6 ft. 9 tons. Explosives 1·5 to 1·7 lb. per ton.

Another thing miners want teaching is that the front holes of a bench should have more ground to break than the back holes, as of necessity the back holes must look in. Also they want teaching the use of the machine under the bar. Many miners swear they cannot get the machine under the bar in low stopes, but if they will start a four-hole bench at the top back hole they will find themselves in good position for turning under when they have finished the front top hole.

In quarrying, it is quite possible to break up to 80 tons from one shift's work and one hole. When breaking bottom, unless there is a very good parting underneath, all the holes should be lifters and drilled below the whole of the reef; I have often seen men breaking bottom where the value has been in the lowest band of reef, with the machines on top drilling vertical holes, thus breaking all the waste and leaving three parts of the pay reef, whereas, with waste on top of reef, lifting holes leave the waste in good condition for sorting. In starting to quarry up the footwall of a block of ground a loose end should be first formed by cutting a gap through the block (Fig. VI.), and the holes then set off at an angle to the dip: it is the same with roofing, get a gap through first and it is then much easier to carry on the work systematically.

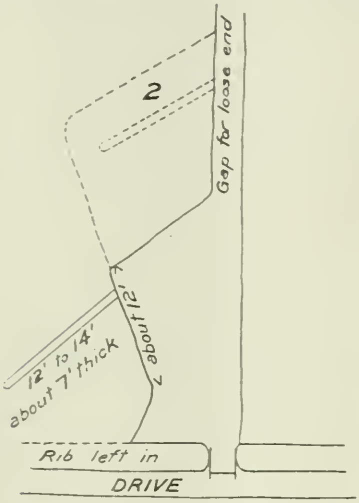


Fig. VI.—Breaking bottom or roofing.

Miscellaneous.—Drill Steel.—Many managers are worried a great deal about the handling of the drill steel, especially where they have skips only, as a great deal of hauling time is taken up in handling drills. In the outcrops, with hammer boys, there is little trouble, as the boys carry their drills in and out with them.

I expect that soon we shall have several detachable bits on the market, and the probability is that one or two will be sold at a figure which will compare well with present cost: they should be served out to the men as stores and charged for, or there will be great waste, enough to prevent their use in some cases. In the meantime the expense of sharpening and handling could be kept down by using as few star bits as possible, for if the ground is fitchery in one part of the mine and needs star bits, there is no reason to fit out every machine with more star bits than necessary, for, as a rule, the starter is the only star bit needed. Star bits cost more for handling and sharpening, and then there is the welding, which is a heavy item. Drill steel should be bought in lengths suitable for cutting up without waste.

For chisel bits, I like the grooved steel as the strength is under the corners, and I used to have the $1\frac{3}{4}$ in. steel, made to carry a 2 in. bit and $1\frac{1}{2}$ in. steel a $1\frac{3}{4}$ in. bit. Men who were new to the use of this grooved steel were ready to believe that a lot of holes would be lost owing to its flat side, but in actual work the steel came out well, although there was only a $\frac{1}{4}$ in. gauge on the drills. I also tried to keep the bit parallel for 1 in. from the cutting edge, and found it of great service in avoiding holes being lost through drills losing gauge (Figs. VII. and VIII.).

For hand drills I found the $\frac{7}{8}$ in. grooved steel did very well, and being much lighter than ordinary $\frac{7}{8}$ in. octagon there are more drills cut from a ton of steel. The boys also did not have so much weight to carry, and the drilling did not suffer, as it was much easier to start holes with the light steel. I had the heads tapered down to about the size of a sixpence, less if anything, and tempered, which saved more than its cost in steel; the hammers were made with the faces not larger than a penny piece, and I experienced very little trouble in breaking in boys to hammer work, though it was predicted that the boys would knock their hands to pieces by using the small faced hammers and small headed drills.

To save handling, furnaces could be put in underground as near the work as possible: at any place where twelve machines are working a furnace could be put in, for there is not much saving in actual cost of handling if the furnaces are put close to the shaft.

Vertical compressors are another little worry, I have always found that they give much more trouble than horizontal ones, and have not been able to find out why so many have been bought. One would think ground was very scarce out here, or may be it was thought that the exercise of climbing about them and the contortionist work necessary when repairs had to be done was beneficial to the men who had to do them. Horizontal engines cost less in upkeep, and a man can look after more, but what I think is the main thing is that there are fewer stoppages with them, and if the loss in the mine due to stoppages was booked against the compressors, I fancy there would not be so many vertical ones ordered.

Turbines.—Something might be done by using the exhaust steam of our high pressure winding engines in low pressure turbines, or mixed pressure turbines. At a colliery in England a plant was put in to deal with the exhaust from the winding engines consisting of, what might be termed a steam accumulator, a low pressure turbine and a generating set. The accumulator was a box or tank in which about 20 tons of old rails, etc., were placed; the hoisting engines exhausted into the accumulator during the time they were running; the turbine was taking steam all the time, and while the hoisting engines were working fairly regularly the accumulator would give off steam to keep the turbine going, but if the hoisting engines had to be stopped for too great a period arrangements were made to supply live steam from the boilers. It was found that there was little alteration in speed during the ordinary intervals between trips, the heat going into the rails, etc., in the tank being given up to the water during the intervals between the hoists.

In speaking of steel, I made no mention of the costs. I have always used cheap steel, costing from $2\frac{1}{2}$d. to $3\frac{1}{2}$d. per lb., and have refused to buy dearer steel for the reason that the so-called cheap steel did the same work as the dearer steel. I have always been of the opinion that the higher priced steel did not get the treatment necessary, and that the treatment that drills got from the sharpeners was more suitable to the cheaper steel. The grade of steel I have been using will stand up to its work very well if the bits are well formed, for the forming of the bits has as much to do with steel being suitable as anything else. The wings of star bits should be hammered out from the centre to bring the metal under the corners instead of the corners being hammered thin and the centre left thick, for if the sharpener is allowed to peen out the corners of a bit, it is bound to fail, no matter what steel is used. I would advise managers to try the effect of taking $\frac{3}{4}$ in. out of the centre of star bits (Fig. VIII.), the solid part, and try its effect on the cutting power of the drills, also to cut a V in the chisel bits, a $\frac{1}{4}$ in. to start with (Fig. VII.).

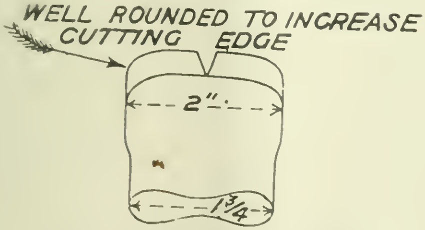


FIG. VII.—Chisel-bit of grooved steel.

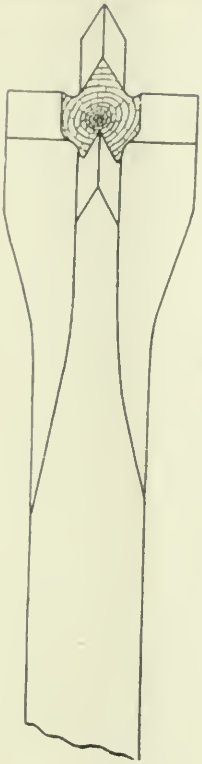
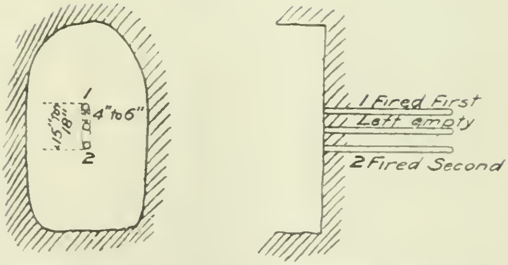


FIG. VIII.—Star-bit. Solid centre drilled or cut out $\frac{1}{2}$ in. or $\frac{3}{4}$ in.

If dearer steel is used it must get better treatment, then it might pay. The sizes of drill bits I used for large machines were: firsts, $2\frac{1}{4}$ in.; seconds, 2 in.; thirds, $1\frac{3}{4}$ in.; fourths, $1\frac{1}{2}$ in.; seconds and thirds were grooved steel, broken from the bars full length. The bits of chisels were dollied and cowmouthed (Fig. VII.), that is, the corners brought back to get more cutting edge on them, where it is wanted.

New Cut for Drivages.—Just before closing down on the South Nourse I was trying a new cut, which I think is worth following up. It was of three holes, but the three holes were in a vertical line and parallel (Fig. IX.); I tried them in a vertical line for ease of drilling, as it only required the arm dropping 6 to 8 in. or so to start the next hole after drilling the first. The holes are simply put in as if for channelling: the middle hole is not charged but left empty, the top and bottom holes are charged and fired one after the other; it makes a gap about 15 to 18 in. high and the width of the hole: it takes less explosive and any practical length of cut can be broken, for so long as the holes are parallel the cut can be broken. As most of you know, if cuts of good length can be brought out, longer rounds can be broken with the same amount of explosive as most miners use at present for the shorter rounds, therefore the cost per foot will be less. From what I saw of the few rounds I tried I have faith in its success. The men would probably object a little at first, because it is a little troublesome to get into the habit of drilling the three holes parallel.



Front elevation. Longitudinal section.

FIG. IX.—Channelling cut.

Guides.—Shaft guides should be fastened with bolts of square section, except the threaded part, and cup heads. In the case of small wooden guides I have often seen nearly the whole of the face cut away at the bolting part to make room for the bolt head and box spanner used to stop the bolt from turning, but bolts of square section driven into a round hole, the same in diameter as the side of the bolt, would hold themselves, and the cup heads would take up little room. Another thing, square nuts should be used instead of hexagon, which may sound a little old fashioned but those who have had trouble in trying to get off tight hexagon nuts, whose flats have become circles, will readily understand why I prefer square nuts: cranked doubled ended spanners should also be used. If the ordinary round coachscrews had to be used in wooden guides in vertical shafts, I should prefer to fasten a piece of angle iron on to the sets and bolt to that instead of coachscrewing to the sets. In pump work, also square nuts should be used: and it would be a good job if pump makers would try and turn out their pumps with as few sizes of nuts as possible, even if they had to put a large nut on a small bolt.

All mines should be fitted with rescue apparatus so that persons could be fetched out of bad places, where they have got gassed, without the rescuer running any risk.

Endeavours should be made, especially by the mine doctors, to get First Aid classes on all mines, but not necessarily under the auspices of the St. John's Society. The companies could offer yearly prizes for the best class work, and it would be a good assurance premium for them.

Cornish Pumps.—In putting Cornish pumps in there is generally too much of the shaft taken up. The pumps themselves should be put in a chamber off the shaft, so that there would only be the rods and column taking up room in the shaft, and T balance bobs should be connected to the back of the rods by a link so as to save room. In many shafts there is 2 ft. or more in the length taken up by the pumps, that ought to be available for something else or permit of the shorter shafts; this applies to both vertical and incline shafts.

Comparison Sheets. — Many members are acquainted with the comparison sheets that the different groups issue, showing costs, etc., on their mines; in many cases these sheets are very useful and in other cases they are misleading, as conditions vary so much. Coal, for instance, is compared on the tons milled, without reference to how much rock is broken by machines or hammer boys, amount sorted, distance hoisted, and number of engines and boiler stations.

Again, as far as explosives are concerned, in one mine the stoping width may be much narrower than in another, but they are still compared on cost per ton produced: say, one mine has a stoping width of 60 in., another 40 in.; tons per fathom, 15 and 10; the 15 tons per fathom mine would come out at about $2\frac{1}{2}$ fathoms per 50 lb. of explosive, or about $1\frac{1}{3}$ lb. per ton, the 10-ton mine would come out at about 2 fathoms per 50 lb. of explosives, or about $2\frac{1}{2}$ lb. per ton, this is where there is no parting at foot or hanging in either case, yet these two would be compared per ton without reference to their stoping widths. As regards explosives, I think a little money could be saved by using more gelignite in place of gelatine, as in most places gelatine is only a cover for poor holes: I fancy this will be considered heresy, but it is nevertheless true.

Another thing in comparisons is the cost per ton stoped. We have one mine producing 11 to 12 tons per machine shift: another about 6 tons; this is not because in the first case there are better miners, but it is a matter of stoping width. An 8 ft. stope will give 24 tons per fathom, a 4 ft. stope 12 tons per fathom; also in the 8 ft. stope a larger fathomage per machine can be broken, yet the cost per ton is compared without reference to the stoping widths. When it is understood, that a 6 ft. stope should turn out approximately twice as much rock per machine per shift as a 4 ft. stope, it is readily seen that a bare comparison of cost per ton is a poor consolation for the manager with the smaller stoping width. In shovelling also we have great differences; one mine may have a steep dip or a smooth footwall, which saves shovelling, another mine may have a rough footwall or a medium dip causing a large amount of shovelling, so that here again bare comparisons are not fair.

This comparison system tends to keep up costs, for in a mine with wide stopes the ground can be broken much cheaper than with narrow stopes, and so long as the manager can keep ahead a little in his costs, he does not worry much. We all know there are certain mines along the reef that were well known as money making mines amongst the miners. Why? because too much money was being paid for the work, and if men can get as much for breaking a unit of easy ground as for a unit of hard ground, they will go for the easy. These money-making mines have helped to keep up wages, for the harder mines have had to compete for the men and, of course, have had to pay more than they should have done. To be fair to all managers, it should be made compulsory for every manager to visit each of the other mines of the group once per quarter, the visiting manager to take special note of anything that is being done cheaper than at his own mine, and to find out whether it is due to circumstances or management. If he did not see for himself he would always put it down to circumstances, of course. Once per month the chief should cause all the managers to meet at his office to have a quiet consultation on work in general, and encourage them to bring forward new ideas.

In case of new work, such as a new mine to be opened up and equipped, he should lay the proposition before them and get their ideas of the different parts of the work, giving them time to think out the thing before deciding; in this way a little more use would be made of the managers, which should result in a saving to the controlling firms. Again, managers and heads of departments should be forced to take a short holiday of a few weeks per year. The beneficial results to the companies would be greater than the costs, as a man cannot do justice to himself when working to a standstill. When taking their long holidays they should be encouraged to visit other fields to see how other people do things, and to see if there is anything that could be introduced to advantage here.

Labour.—I think the reason why the employment of more white labour is not taken in hand is because many people do not look for the causes of failure and remedy them. I am of the opinion that more white labour can be used if the men in charge would give themselves a little trouble in trying to make it a success. If whites can be found to do the work as cheaply as natives then I consider it the duty of men in charge of work to do their best, even at a little extra trouble to themselves, to get whites into work. If the base metal industry is to grow as it should, there are not natives enough to go round, and the greater the scarcity, the higher the wages will have to be: why not try therefore to keep down the native cost by using whites! It may sound a hare-brained proposition, but it is not so wild as it looks, as I know from personal knowledge.

In machine work it is not tons per man that is to be looked for only, it is cost per ton: if work can be done at a certain figure per unit by one man running three or more machines, then try to get it done at the same price by putting one man on each machine. The companies could afford to pay more really as there would be a saving of coloured labour and a greater efficiency per machine.

Of course, it will be said that we have not machinemen enough; that is so at present, and we never shall have, if each man is to look after four or five machines, for he has not time to learn his work; but put eight or ten machines in a stope, a white man on each with a helper drilling at a per foot rate, and a contractor at a per fathom rate for the whole, or a shiftboss working on wages, and bonus for fathoms broken and explosives per fathom, then there would be a chance of making a few machinemen. Also in shovelling a few whites could be got in without increasing costs. I am not proposing white labour for everything, but would ask managers to study closely the cost of different work and then see if they could not do as well with white labour as with coloured in many things which coloured are now doing. I know the consensus of opinion is against me on this question, but having been using labour at many kinds of work for over 30 years, I have had time enough to form an opinion for myself.

There is one thing that, to my mind, always seems to be forgotten, and that is that we require a much better average class of miner in this country than anywhere else where white labour is used: we have no room with the present system for anything not of the best. As to the three years' system of training, I do not think it suitable for the general run of men; it is a good thing for mining students who are preparing themselves for holding official positions; but for the ordinary workman, I have found that a great proportion of the worst men we have are amongst old miners; so with accidents, the great proportion of fatal accidents occur to old hands. The man who cannot learn to break ground in these mines in six months will never make a miner. I do not think it a good policy to try and make every man fit to take any job underground; there is nowhere else that it has been attempted that I know of. We want a system that will allow the lads to come in, not boys of 18; boys of 18 are the men of these days with families of their own, and we talk of putting them as apprentices to mining; if we are to help the youngsters, we must not wait until they are 18, as at that age they should be ready to take their share of the burden as men.

There is one thing certain, if the white labourers wish to be largely employed in these mines they must be prepared to get down to solid work; it is getting time they realised that the happy-go-lucky days are over, and if they want employment they must make it pay for the mine people to employ them. If they go to any other country they will have to work hard, so they may as well make up their minds and start in right here.

There are a few things in this paper that have already been dealt with by this Society, but there is still room for further discussion which, whether the writer is correct or not, is always a useful thing.

The President: I do not know whether Mr. Johnson set out with the deliberate intention of introducing all the contentious points in mining into one paper. If he did, he certainly succeeded very well. As far as I can see, the numerous points raised in the paper could very well keep the Society discussing them for months. There are two points in this paper which I would just like to mention to-night. One is the author's reference to Kimberley, in connection with hoisting 2,300 tons from a depth of 1,200 ft. in the course of eight hours. He says that he cannot see any reason why the same good work cannot be done here. I would point out there is rather a difference between the conditions here and the conditions at Kimberley. Take De Beers, you only have the one hoisting station; it serves the whole mine, and everything is dropped to that station. On the Rand we have a different state of affairs, for we have vertical and incline shafts, and because of the latter we have, say, as a rule, a loading bin at every other station. That being the case, it means we have quite a number of loading stations in our incline shafts, and we cannot afford to go to the same expense on each station as they do at Kimberley. The number of our loading stations naturally interferes with the speed of hoisting. Another point which I picked up in the course of reading the paper was in connection with telephones. It has often occurred to me that we might very advantageously introduce telephones, in our deep levels at any rate. I have personally lost many an hour underground hunting for shift bosses and others, and they have lost hours looking for me. The telephone is quite cheap, and I am in agreement with him on that point. I move a very hearty vote of thanks to Mr. Johnson for this particularly interesting paper.

Mr. T. Lane Carter: I have much pleasure in seconding the vote of thanks to Mr. Johnson for his lively paper. There is so much that is contentious in this contribution that I predict it will bring out all the fighting spirit that we have in this Society. As a matter of fact, I believe it will be as conducive to discussion as the paper given by the Messrs. Denny on the "New Metallurgy."

I wish there were time to pitch in to Mr. Johnson to-night, but time is pressing and I will not begin. With some of his ideas I find myself in agreement, but with other statements my views are opposed to those of the author.

Mr. Johnson must have given much time and thought to the preparation of this paper, as it covers so much ground. I have enjoyed listening to the reading of it and, I am sure, you will all agree with me that the author deserves our thanks. At a future meeting I hope to open fire on some of Mr. Johnson's positions.

RESCUE APPLIANCES: LESSONS FROM GLENCOE.

By H. Kestner.

This paper was read by the author and illustrated with a carefully prepared diagram on the blackboard, but owing to pressure on space this paper is postponed to the next number of the *Journal.*

The President: I may say that I heard that Mr. Kestner was down at Glencoe, and it occurred to me that it would be a very opportune time for him to give us a paper describing the "Shamrock," and also giving us his experiences at Glencoe. You will remember that it was the "Shamrock" apparatus which did such yeoman service in the terrible disaster in the north of France, and I have reason to believe, quite apart from what Mr. Kestner has told us, that the "Shamrock" has been doing good work at Glencoe. I am glad that he has promised to bring the apparatus for our inspection as soon as it arrives. There is only one in the country at the present time, but some have been cabled for, and we shall have an opportunity of seeing one. I move a vote of thanks to Mr. Kestner for his valuable paper.

SOME FEATURES OF SILVER ORE TREATMENT IN MEXICO.

By W. A Caldecott, B.A., F.C.S. (Member).

Part II.*—Theoretical Considerations.

The reactions involved in the dissolving of silver from its ores admittedly still require much study before they are clearly understood. The following discussion is rendered imperfect on account of the non-arrival of a parcel of Mexican silver ore, upon which it was intended to carry out tests to develop and confirm the tentative conclusions arrived at from an examination of the properties of artificially prepared silver sulphide. The study is instructive, as apart from its direct interest, it has a bearing upon the treatment of gold ores containing sulphides which dissolve in cyanide solution. It is probable that since the mineral silver sulphide is coarser and more compact than the artificially prepared compound its dissolving in cyanide solution is slower, thus allowing more time for secondary reactions to proceed.

The primary reaction in the dissolving of silver sulphide from its ores may be taken as:—

$$Ag_2S + 4NaCy = 2NaAgCy_2 + Na_2S \qquad (1)†$$

When, however, a certain amount of silver has been dissolved by the above reaction an equilibrium is established and solution ceases. How far the reaction has proceeded before this equilibrium is attained, depends on the amount of soluble sulphide in solution, whether derived from silver sulphide or other sources, and the amount of free cyanide present.[*]

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* Part I., on Works Practice, read at January Meeting, 1908.

† Cp. equation $Ag_2S + 6KCy + H_2O = 2K_2AgCy_3 + KOH + KSH$, suggested on p. 241 of this *Journal*, March, 1905.

This is illustrated by the following table, showing the results obtained from a series of argentiferous sodium cyanide solutions of the strengths stated, 100 c.c. of each of which was titrated with 0·1 % sodium sulphide solution until a brownish colouration was just produced. A deduction of 0·05 c.c. was made from the reading, so as to give the amount of sodium sulphide that could be added without reacting with the silver present.

Table A.

Per cent. free cyanide (as KCy)	Silver present mgrms per litre.	Na_2S required mgrms. per litre.	Equiv. sol. Ag. present (as Ag_2S) mgrms. per litre.
0·50	490	5·3	14·7
0·40	392	4·4	12·2
0·30	294	3·1	8·6
0·20	196	2·0	5·5
0·10	98	1·0	2·8
0·05	49	0·7	1·9

An inspection of the foregoing table, in which milligrams per litre may be read as grams per metric ton, shows that the amount of silver sulphide capable of being dissolved as the soluble double alkaline cyanide is approximately proportionate to the amount of free cyanide present, but even with 0·5 % solution the quantity of dissolved silver is less than 0·5 oz. per 2,000 lb. ton. Further, it is obvious that the introduction, through presence of a free alkali or otherwise, of soluble sulphides into an argentiferous cyanide solution from any mineral or other source will result in the precipitation of silver already dissolved. The fact that dissolved silver and other than very small amounts of sodium sulphide cannot exist simultaneously in the same solution serves to explain why working solutions do not show more than traces of the latter when sulphide ores are being treated. With solutions low in silver it is difficult to filter off the very finely divided suspension or coloured colloidal precipitate formed on addition of sodium sulphide solution to sodium argento-cyanide solution.[†]

The explanation of how in practice many ounces per ton of silver existing as sulphide in the ore can be dissolved can be explained by the occurrence of secondary reactions, and to these attention may now be directed. They show the conversion of soluble alkaline sulphide first formed into other compounds which do not preclude the further solution of silver, and thus the primary reaction appears to be merely a transition state in the sequence of reactions. No doubt, several of these secondary reactions proceed simultaneously, and the formation of the resultant products indicated has been experimentally proved by their detection in the solution.

The following are the secondary reactions referred to:—

$$(a)\ \begin{cases} 2Na_2S + 2O_2 = Na_2S_2O_3 + Na_2O & (2) \\ Na_2S_2O_3 + Na_2O + 2O_2 = 2Na_2SO_4 & (3)^* \end{cases}$$

$$(b)\quad Na_2S + NaCy + O = NaCyS + Na_2O \qquad (4)$$

$$(c)\ \begin{cases} Na_2S + PbO = PbS + Na_2O & (5) \\ PbS + NaCy + O = NaCyS + PbO & (6)^\dagger \end{cases}$$

The foregoing reactions, when carried to completeness, all involve absorption of oxygen, whereby the utility of aeration may be accounted for, apart from its accelerating the solution of any metallic silver or gold present in an ore under treatment. Reaction (*a*) apparently proceeds a good deal more rapidly than (*b*), and since it does not involve destruction of cyanide, and indeed it is not required to proceed further than the first stage, resulting in the production of sodium thiosulphate (itself a silver solvent), this may be considered the most desirable result of aeration. Reaction (*c*) with lead present as suspended hydrate or soluble alkaline plumbite has the advantage of instantly converting the soluble into insoluble sulphide, and also, the reaction being cyclical, given sufficient oxygen, a limited quantity of lead salt serves to remove an indefinite amount of soluble sulphide, but this last result is only attained at the expense of free cyanide being converted into useless sulphocyanide. If lead peroxide could be employed as soluble alkaline plumbate, a double quantity of soluble sulphide should be removed as sulphocyanide and insoluble sulphide.

To ascertain whether, in addition to the foregoing reactions involving absorption of dissolved oxygen, sodium sulphide could be oxidised at the expense of oxygen combined in water, an aqueous 2·97 % solution of sodium sulphide was allowed to stand for five days in a filled and closed vessel. At the end of this time no loss of strength could be detected, nor were thiosulphates nor sulphates present, whereas a parallel test in a closed flask only one-eleventh filled showed on opening reduction of internal pressure, a reduction of strength to 2·85 %, and the presence of sulphate in solution.

Since in Mexican practice little lead seems to be deposited from solutions entering the zinc boxes it does not appear as if the cycle (*c*) were usually completed, but from a scientific standpoint its possibility is of interest. Apart from any re-formation of PbO, nearly as much lead as there had been silver present as sulphide is required to carry equation (5) to completion.

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[*] Heat also increases solubility of Ag_2S in cyanide solution. See Percy "Metallurgy of Silver and Gold." Part I., p. 21, 1880; also Louis Janin, *Eng. and Min. Journal*, Dec. 29, 1888, and *Min. and Sc. Press*, July 2, 1892.

[†] See Watts' Dictionary of Chemistry, New Edition, p. 472, vol. iv.

[*] See Watts' Dictionary of Chemistry, New Edition, p. 483, vol. iv.

[†] See Bettel & Feldtmann, this *Journal*, vol. i., p. 209, 1897.

Whilst the relatively weak solutions included in Table A dissolve silver sulphide in proportion to the total amount of cyanide present, yet with concentrated solutions this does not hold.* A strong cyanide solution saturated with silver sulphide will yield a deposit of this salt on dilution with water, and such a precipitate can be successively formed and dissolved by alternate additions of water and concentrated cyanide solution.

A study of the molecular weights involved in equation (1) shows that the silver requires nearly its own weight of sodium cyanide to dissolve it, so that the consumption from this cause alone with 500 gm. silver ore would be nearly a pound of sodium cyanide a ton. This feature, as one of the sources of high cyanide consumption experienced with silver ore, renders the field for cyanide regeneration processes a good deal more promising than with banket gold ore, where the total cyanide consumption need not necessarily exceed a very few ounces per ton. Apart from reactions taking place in the presence of lead salts, the absorption of oxygen in reactions (*a*) and (*b*) is considerable: in fact, even in the stage of (2) it amounts to one-seventh of the weight of the silver dissolved. Hence, under these conditions, although oxygen does not play the direct part in dissolving the silver that it does in gold ore treatment, yet its importance in permitting solution to proceed, and the present necessity for the tedious process of repeated agitation and aeration, appear evident. On like grounds experiments by agitating silver sulphide and cyanide solution with the oxidiser barium dioxide, or with air under 100 lb. pressure, showed considerably more silver dissolved in a given time than in the absence of either. To illustrate the fact that introduction of sodium sulphide does not cause precipitation of silver when sufficient dissolved lead is present, the following table is of interest, since from this it follows that presence of dissolved lead allows solution of silver sulphide to proceed, as in equations (1), (5) and (6).

A solution of lead plumbite containing 1·95% PbO was prepared by dissolving precipitated lead hydrate in caustic soda solution. Six argentiferous sodium cyanide solutions, of the strengths stated were also prepared, and to each added the stated amount of lead as plumbite. Then to each was added sufficient sodium sulphide in solution to combine with all the dissolved silver present. The flasks containing the solutions were then closed and agitated for an hour, after which the solutions were filtered and the precipitate assayed for silver. The results are as follows:—

Table B.

Per cent. free cyanide (as KCy).	Silver present mgrms. per litre.	PbO added as plumbite mgrms. per litre.	Na_2S added mgrms. per litre.	Mrgms. silver precipitated per litre.
0·50	490	2,600	177	3·37
0·40	392	2,080	141	1·73
0·30	294	1,560	106	2·00
0·20	196	1,040	71	2·30
0·10	98	520	35	0·90
0·05	49	260	18	0·37

The trivial amounts of silver in the precipitate show how effectual is the presence of lead in preventing precipitation of silver, and consequently in permitting the dissolving of silver sulphide to proceed in spite of the concurrent formation of soluble sulphide. To confirm this latter deduction, the first and last experiments of Table B were carried out again on similar lines, but in this instance the sulphide was added before the plumbite. Only trifling amounts of silver were found in the precipitate.

Mercury salts probably act in a similar way to lead salts in forming an insoluble sulphide, though but little used on account of their higher cost. Possibly also direct replacement of mercury in the double alkaline cyanide by silver takes place, as claimed by Keith and Hood.* It is claimed that only mercury in the mercurous state assists in the dissolution of the silver, so that use of calomel should be preferable to that of mercuric chloride.†

To account for the dissolving of metallic silver and silver chloride, the following simple equations may serve:—

$$Ag_2 + 4NaCy + O = 2NaAgCy_2 + Na_2O \quad (7)$$

$$AgCl + 2NaCy = NaAgCy_2 + NaCl \quad (8)$$

In order to determine how far results in practice are affected by chemical reactions, and how far by the mechanical factor of fineness of crushing of the silicious matrix, it may be of interest to recall a method of examining cyanide residues for encased values.‡ In examining residues of silver ore, boiling in strong nitric acid might be employed followed, after washing, by agitation with strong cyanide solution and barium dioxide. Boiling in aqua regia in the first instance might tend to produce insoluble silver chloride, whereas nitric acid would form soluble silver nitrate, and after its removal any exposed gold would be dissolved by cyanide as above, or aqua regia might then be used for the second stage.

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* See Comey's Dictionary of Solubilities, p. 375.

* See Rose's Metallurgy of Gold, 1902, p. 405; also this *Journal*, p. 326, April, 1906.

† See this *Journal*, Feb., 1907, p. 272.

‡ See this *Journal*, vol. iv., July 1903, p. 110.

In conclusion, I trust that this brief and imperfect account of some metallurgical conditions and problems in the largest silver-producing country in the world* will be considered as a timely variant from our ever-present problems of gold ore treatment. If the publication of this paper should serve to elicit further information from those engaged in treating silver ores, or from those interested in the subject from its scientific aspect, the author's object will have been attained.

I desire to express my indebtedness to Messrs. A. McA. Johnston and F. W. Watson for carrying out at the Consolidated Goldfields laboratory the experimental work quoted in the paper.

DISCUSSION.

Prof. J. A. Wilkinson: The precedent which Mr Caldecott has set by giving us the benefit of his experiences in metallurgical work whilst on a scientific holiday, so to speak, is one which in the history of this Society is unique, but I think it is one which is well worth of imitation by those who are fortunate enough to be able to undertake them. It is to be hoped, therefore, that the worthy example which he has set will be the predecessor of many which will contribute noteworthy additions to the metallurgical literature of our Society and in thuswise enhance the value of our Journal, not only to our own members but to others. There is another phase of this which must not be lost sight of, namely, that in studying problems dealing with metallurgical ores and products vastly different from our own, we may often glean facts, which have a bearing on our own methods of treatment, or from analogies we may be led to adopt points of perspective which otherwise would have been lost to us, but which on the other hand may lead us to the correct solution we had been vainly endeavouring to find. We are therefore indebted to the author of this paper for riveting our attention on a process so like our own as to be almost a slavish imitation, but one which yet calls for skilful handling and much scientific work.

According to the dates given, the birth of the patio process dates back 350 years; as far as age is concerned therefore, there are few metallurgical processes, if any, with such a record. It would be very interesting if Mr. Caldecott would give us some details as to its working at the present day, and state whether the process differs in any essential from the accounts given in ordinary text books. The severe competition of more modern processes generally induces in older methods alterations which tend to make them more effective and efficient, and in this respect possibly those who are conservative enough to keep to this ancient method, have introduced changes, which will make this process "die hard." Also to what extent are the two processes, the patio and the cyanide, used at the present day in Mexico? Further, does the author know of any scientific explanation of the chemical reactions occurring in the patio other than the suppositions ones given in text books? This has been, I am aware, a very much debated question, and I have ventured to ask this in order to ascertain if there is at the present day any agreement between different observers.

It is pleasant to hear from one who has had such recent experiences as the author, confirmation of the current reports with regard to the good government of the country under the guidance of President Diaz, who from all accounts is one of the ablest administrators the country has ever possessed. Labour troubles, such as are our *bête noir* here, seem to be unknown, and there is doubtless a future for the country, such as any one reading between the lines of the author's remarks on this subject can easily predict. As regards the cyanide process itself, it is so new that even modern text books give no account of it, and hence we must acknowledge our further indebtedness to the author on this score. As regards the ore, I venture to assert, it would have been better if something more than a general statement had been given. Also what percentage of concentrates is obtained and what generally speaking is their composition? Could these not be economically treated along with the general bulk of the ore by finer crushing? It would also be interesting from the chemical point of view to know in what condition the arsenic and antimony is contained in the ore, whether in combination with the silver or otherwise.

The author in his paper has given us a bold outline in general terms of this new method of extraction, and to supplement this and at the same time corroborate his accurate observations I have prepared a rough flow sheet from a description of the process, as carried out by the Guanajuato Consolidated Mining and Milling Co.,* which is appended.

The ore used in the above mill is stated to contain 85% silica, 4—5% iron, and 25 dwt. of gold and silver, 10% of which is gold and 90% silver. The concentrates from the tables and vanners, which is sent to the smelters contains about 50% iron sulphide (pyrites?) and 5% silica. In the pulp thickening vats the slime is thickened to 65% moisture, and then undergoes agitation with 1% cyanide for 24 hours. Lime is added to the pulp in the batteries and another

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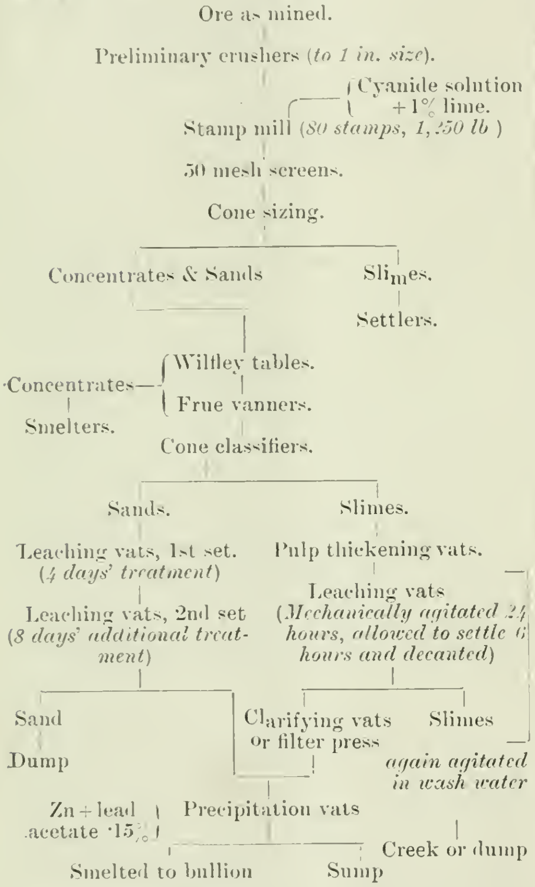
* During 1906 Mexico yielded 68·5 million ounces of silver, or 37% of the world's production. *Mineral Industry*, p. 343, vol. xv

* *The Mining and Scientific Press*, Jan. 25, 1908.

1% to the sand, when transferred from the upper to the lower vats. Lead acetate is also added, ·05% to the sand and ·15% to the slime. The consumption of cyanide is 2·86 lb. per ton of ore, and zinc 1·37 lb. per ton of ore treated. The stamp duty works out at 3·75 tons per stamp per day of 24 hours. Extraction is stated as 90% Ag and 97—98% Au.


Ore as mined.
Preliminary crushers (*to 1 in. size*).
Cyanide solution + 1% lime.
Stamp mill (*80 stamps, 1,250 lb*)
50 mesh screens.
Cone sizing.
Concentrates & Sands
Slimes.
Settlers.
Wiltley tables.
Concentrates—
Frue vanners.
Smelters.
Cone classifiers.
Sands.
Slimes.
Leaching vats, 1st set. (*4 days' treatment*)
Pulp thickening vats.
Leaching vats (*Mechanically agitated 24 hours, allowed to settle 6 hours and decanted*)
Leaching vats, 2nd set (*8 days' additional treatment*)
Sand
Dump
Clarifying vats or filter press
Slimes
again agitated in wash water
Zn + lead acetate ·15%
Precipitation vats
Creek or dump
Smelted to bullion
Sump

NOTES ON SMALL STOPE DRILLS.

(Read at October Meeting, 1907.)

By E. M. Weston (Associate).

DISCUSSION.

Mr. M. H. Coombe: Mr. Weston has again done good service to the industry by bringing forward his second paper on rock drilling machines. What a storm of controversy is raging around the vexed points of this discussion at present! It makes one feel timid of entering the arena because, willy-nilly, one is bound all unwittingly to tread on the tail of some one's coat. One remarkable feature of the revived interest in rock drills is the number of people one meets who could not drill a round of holes with a machine to save their lives, and yet they talk glibly and learnedly about valve motions, and reciprocating versus hammer types, and large versus small, and machines versus hand labour, and discuss machine men and their ways, and pronounce judgment generally on a large and vital subject of which they know absolutely nothing. Then, again, there has been quite a lot of clever advertising. The Gordon people managed that part of the business very nicely, whilst the makers of the "Little Wonder" came a good second, and at our January meeting the "Konomax" got a very good free advertisement. Now I have been wondering what the man who stands at the back of it all has been thinking of all the fuss. I mean the operator, the machine man, the brain and the arm, without which the machine is of no avail. I am pleased to see that our President, Prof. Yates, in his most instructive paper on the recent tests lays stress not infrequently on the importance of the man behind the machines. The operator is all important and knows more of the capabilities of the divers makes of machines than the makers themselves. He knows far more than the average manager or mine foreman of what a machine should do, and yet his advice is but seldom asked and his counsels when tendered are ignored. He told his manager years ago that air pipes were too small, and consequently the friction too great, and air pressures at machines too low for economical work. The mechanical engineer said he was a fool. He also pointed out that the drills served out to him were too few in number, were badly sharpened and tempered, and broke off or bent too frequently to allow of his doing good work. He was told he was not a judge of steel. He asked for a jet and they gave him a spray or an atomiser or some other "fool device" which helped to increase his speed in the direction of Braamfontein cemetery. He asked to be allowed to sink winzes instead of murdering himself in a rise and was told he was not paid as consulting engineer. In fact, he asked for most of the things that are now considered new departures, and which, if his views had been listened to and acted upon long ago, would have saved a good few thousand pounds to shareholders. Now he has his opinions about these tests and the rock drill questions that are agitating our minds to-day, and, as I think I know them, you may perhaps be interested to hear what they are.

In the first place, he has no use for small machines. Despite the tests and their results, he is firmly of the opinion that in a small stope of 36 in. or under no mechanical appliance can compete with hand labour. On a basis of seven boys equal to one small machine, and a choice of contract with either hand labour or machines, he would choose the boys, maintaining he would come out better at the end of the month and with less cost to the company. Were he compelled to take machines then he would choose a certain maker's 3¼ in. drill. I am not going to make this an occasion for a free advertisement: it is a well-known machine, and is undoubtedly the mark at which all rock drill makers are aiming. His arguments are that he would have less rigging up, less hoses to connect, less pipes and connections to look after and keep in order, and far less running up and down the stope looking after "fitchered" holes, broken machines, drills, etc. He claims he would do more drilling and keep his stope benches in better order and thus break more ground. Giving a man five small machines is economically as bad as giving him three large ones to look after. The man behind the handle cannot understand the arguments put forward in the attempt to justify this course from an economic standpoint. He knows certainly that one-man-one-machine is true economy in rock drilling, and knows just as well that a vital economical mistake would be made if, supposing the Gordon or any other small drill were adopted for small stopes, unskilled whites or natives should be put to run them. Take an unskilled white at 10s. a shift. An incompetent man is dear at any price and at any kind of work; how much more so at work with a costly piece of machinery using the most expensive driving power that can be supplied him! Then there is the all important factor of breaking the ground. The smaller and tighter the face the more skilful must the miner be to get good results, and as one miner cannot possibly do justice to five machines, or even to two, it follows (in the miners' opinion) that to get the best results, and consequently the greatest saving in costs, it is needful that one skilled man operate one machine only.

On the whole, he thinks there will not be much demand for the small stoping drills. There are two principal reasons that would justify their introduction: but these, in his opinion, will not find an application on this field. They are, first, shortage of unskilled labour; second, cheapening of working costs. Although we may shortly be mourning the absence of the Chinaman, yet it is the miner's firm belief there will be no shortage of unskilled native labour. He says the Government has taken the matter in hand and that they can and will keep up the supply. If that proves correct, then the second condition is disposed of, as hand labour will undoubtedly be found cheaper than mechanical. The solving of the whole question is an abundant supply of native labour. That would settle the machine question for all time, and they, the machines, would be relegated to their proper place in the mine, namely, development pure and simple. Hand labour can beat them badly in stoping and sinking; and in other countries where even white men at a high wage are employed as miners, the manager would be deemed mad or incompetent who would put machines into such narrow stoping widths as ours. If hand labour is not forthcoming, and we are compelled to stope with either small or large machines then let us be certain that we have those essentials that will make for success.

In the first place the best men that can be obtained must run the machines. We must have the best machine, without fear or favour, that is on the market, either small or large. The machines must be kept in good order and the men supplied with spare parts as wanted, such spares to be kept underground so that a man need not throw away half a shift going to surface for, say, a chuck bolt, or side rod. There should be plenty of drills and the best of steel, well distributed, sharpened and tempered.

We need powerful compressors, large air pipes, tight joints, and air at 70 or 80 lb. pressure delivered to the machine.

A tremendous waste of power in air pipes is caused by the tortuous winding of the average Rand drive. The friction set up and consequent increased load on the compressor in some cases must be very great. It has occurred to me that a solution of this as well as a great many other difficulties of Rand mining would be solved if, say, every 4th level was driven absolutely straight to line under the reef at a distance that would give the least possible variation from point to point from the reef, making, of course, due allowance for faults, dykes, etc. These straight drives could be utilised for the laying of the main air pipes, the intermediate stopes and levels being served by right angled installations off the main lines above and below. The many other uses to which these drives could be put, such as mechanical haulage, facilities for boxes and the handling of ore (the drives being under the reef the vertical boxes feeding the haulage would have entrance well up in the stope), the facilitating of shovelling, etc., do not enter into the scope of this discussion.

REPLY TO DISCUSSION.

Mr. E. M. Weston: I must thank members for dealing so leniently with my paper and for supplying constructive rather than destructive criticism. I think I will be able to show that the recent drill tests, as far as they went, tended to confirm the position I took up in my paper, regarding various controversial matters. The enterprise of the publication which started these tests is highly to be commended, and I am sure it has thus brought us nearer the solution of the problem.

Before replying specially to various criticisms ask leave to draw attention to several points in connection with these tests that apparently have escaped the attention of other observers. These tests furnished a partial answer to some of the queries I raised in the latter part of my paper, regarding the philosophy of the process of rock drilling. I am pleased also to see that so capable and painstaking an observer as Prof. Orr will be placed in a position to gain and make public other valuable information. I believe that it is only along the lines of careful investigation and experiment in our own rock and under our own conditions that this stoping problem will be solved.

I raised the query as to what was the best ratio of foot pounds of energy to inch width of cutting edge. The tests seem to show that this ratio may with advantage be a very high one in percussive machines. With good steel the limit is reached only when the bits begin to bend and break.

It was evident, however, that the Chersen machine with its 6 in. stroke and $2\frac{3}{4}$ in. diameter piston somewhat exceeded this limit with air of over 50 lb. pressure, and that bent and broken steel resulted. Nothing in the trials is more striking than the great increase of drilling speed corresponding to reduction of diameter of the cutting edge. With the same size cylinder and the same air pressure, provided the hole clears itself, the speed of drilling bears a definite ratio to the size of bit used or the amount of rock excavated. This is well shown by comparing the speed of drilling of the various machines on six wet holes when boring with starter and finisher drill.

Name of Machine.	Size of Starter Bit.	Cubic Inches Rock per 1 in. Drilled.	Rate of Boring Inches per Minute.	Size of Finisher Bit.	Cubic Inches Rock per 1 in. Drilled.	Rate of Boring Inches per Minute.	Percentage decrease of rock cut in smaller hole compared with large.	Increase per cent. of speed in boring smaller hole as compared with large.
	Inches.			Inches.				
Kimber ...	$1\frac{7}{8}$	2·76	1·13	$1\frac{1}{4}$	1·22	2·01	55·8	77·8
Little Wonder	$1\frac{3}{4}$	2·40	·80	$1\frac{1}{8}$	0·99	1·30	58·7	62·5
Gordon ...	$1\frac{1}{2}$	1·76	1·61	,,	,,	1·89	43·7	16·7
Kid ...	$1\frac{1}{2}$	1·76	1·23	,,	,,	1·71	43·7	47·2
Ingersoll ...	$1\frac{1}{2}$	1·76	1·73	,,	,,	3·30	43·7	90·7
Flottman ...	$1\frac{1}{2}$	1·76	·97	,,	,,	1·73	43·7	78·3
Holman ...	$1\frac{3}{4}$	2·40	1·45	,,	,,	2·29	58·7	58·0
Chersen ...	$1\frac{1}{2}$	1·76	1·60	,,	,,	3·63	43·7	127·0

This table shows a remarkable rate of increase in drilling speed. corresponding to reduction of diameter of hole driven, and curiously enough the most powerful machines appear to benefit most.

These facts had, as will be shown later, a most important bearing on the result of the trials.

Regarding the conditions drawn up for the trials, all I can say is, that the makers of the various machines using only solid steel showed a great deal more pluck than discretion in entering for a contest whose conditions seemed expressly designed to prevent any such machine having a chance of winning. I am not expressing any opinion that the tests set did not represent the requirements for a perfect stope drill. Nor do I contend that such a drill should not be able to put in a hole with equal speed in any direction : but I do say that as mining is at present conducted on the Rand 95% of stope holes are down wet holes that can be bored efficiently by machines using solid steel. Flat holes are seldom called for. Wonderful results might perhaps be gained at sŏme future time by adopting the so-called "new system of back stoping" ; but I think I am right in saying that the attitude of the majority of mining men on the Rand to such a scheme is a wise indifference, although all would wish to run stopes more nearly parallel to drives, and other things being equal, they would welcome a machine that could do this. To, however, make the conditions of test the drilling of 50% dry holes underground and 25% on the surface did not truly represent the requirements of the average mine manager for a stoping drill at the present time.

It is hard to understand why the demand was made that on block D all machines were compelled to start their hour's drilling on an absolutely flat hole, which everyone ought to have known could not be finished in much unde ran hour by any machine using solid steel. This arrangement alone sealed the doom of a machine like the Chersen. Had all machines been allowed to start on an inclined up hole some progress could have been made. This arrangement was nearly as unreasonable as that one contended for by a certain agent that the machines should be set to drill a dry vertical down hole. This he claimed represented mining conditions !

I believe that the Gordon machine, taking everything into consideration, was the best machine competing under the conditions of the contest and deserved the prize, though I should expect a machine like the Chersen to put in more holes in a day's work underground in a 36-in. stope than any other in the contest. My remarks in my paper, I think, showed that I appreciated the many advantages of a machine of the hammer type with hollow steel and water feed, and no one has a higher regard than I have for the technical abilities of the designers of the winning drill. One may have doubts whether in the hands of unskilled labour the trouble arising from drills sometimes sticking in the hole from burred up or broken anvil blocks, and from water entering the working parts of the machine from the water feed or from damp air (for the mineral water in our mines has a bad effect, particularly in the interiors of small drills), may not outweigh all other advantages, and also whether the practice of firing long range high velocity projectiles at the head of a drill bit is going in the long run to be the best method of boring rock. Personally, I should prefer to use a heavier hammer with shorter stroke, but every drill must be more or less a compromise, like a war ship.

Since the trials my respect for the stragetic abilities of the Gordon engineers and their knowledge of the art of contest running is unbounded.

An example of these abilities may perhaps throw some interesting light on the published results and on the disappointing performances of certain drills. It will perhaps show the agents of other machines, and particularly those of three machines that they have very much indeed to learn in the competition business.

If we turn to rule regarding "bits of drill steels" we read, "but the finishing bit shall not be less than $1\frac{1}{4}$ in. for 36 in. holes, and $1\frac{1}{8}$ in. for holes over this depth. By this is meant that steels under 43 in. in length over all must have $1\frac{1}{4}$ in. diameter bits." Nothing is, however, said as to the depth to be bored by each length of steel.

Obviously, the intention of the framers of this rule was to provide that the diameter of the bottom of a hole 36 in. deep should be as nearly as possible $1\frac{1}{4}$ in., and that the diameter of the bottom of a hole 45 in. should be as nearly as possible $1\frac{1}{8}$ in. It was, I believe, felt that the usual hand hammer hole was of scarcely large enough diameter at the bottom to break the maximum economic burden, and that machines should be called upon to bore a slightly larger hole than that hand drillers bore, in order that, to quote from the words of a well known expert, "the powder may be put, where it belongs, at the bottom of the hole." This advantage was specially claimed for one machine as compared with hand stoping. Now rules (like acts of parliament and patents) often have a carriage and pair driven through them, and this one, owing to some omission, was no exception. It is evident that most competitors interpreted the rule as above, and set out to try and bore 36 in. as near as possible with a $1\frac{1}{4}$ in. bit. As before stated, they did not realise the influence of diameter of hole on boring speed and they fell in badly. The Gordon engineers, however, knew their business better. They decided to put in a hole of about 41 in., and they further resolved to bore the irreducible minimum length of that hole with bits of $1\frac{1}{4}$ in. diameter and over, and the maximum length with a bit of $1\frac{1}{8}$ in.

Taking an average over the first six completed wet holes, the following table gives food for thought:

It will be seen at a glance that the Kimber, Little Wonder and Holman machines were hopelessly handicapped from the start by the mistake made in selecting wrong sizes of bits. The Kimber machine in particular was set an impossible task, even if the quality of steel supplied had been good. It had to excavate 67% more ground than the Gordon machine!

The comments on the disappointing show made by the Little Wonder machine that were heard

Name of Drill.	Boring Rate on all completed Wet Holes Average Inches per Minute. @ 50 lb.	@ 60 lb.	Average Depth Hole.	Amount of Cubic Inches Rock Excavated, Average per Hole, no Allowance for Wear of Bits	Percentage Depth Hole Bored with Bits $1\frac{1}{4}$ in and over.	Inches Depth Bored with $1\frac{1}{4}$ in. Bit and over.	Percentage of Rock Excavated with $1\frac{1}{8}$ in. Bit.	Sizes of Bits Used in Inches.
Kimber	·89	1·46	40·83	86·778	100	40·83	nil	$1\frac{7}{8}$, $1\frac{3}{4}$, $1\frac{5}{8}$, $1\frac{1}{2}$, $1\frac{3}{8}$, $1\frac{1}{4}$
Little Wonder ...	1·25	1·43	44·33	69·606	78·2	34·69	13·7	$1\frac{3}{4}$, $1\frac{1}{2}$, $1\frac{1}{4}$, $1\frac{1}{8}$
Gordon	1·57	1·89	40·77	52·023	62·45	25·41	29·3	$1\frac{1}{2}$, $1\frac{3}{8}$, $1\frac{1}{4}$, $1\frac{1}{8}$
Kid	1·21	1·89	41·52	57·636	73·65	30·59	20·5	,, ,,
Ingersoll... ...	2·08	2·57	42·60	57·325	69·10	29·44	22·6	,, ,,
Flottman ...	1·70	1·90						
Chersen	2·35	2·94	42·96	58·275	58·275	32·83	17·6	,, ,,
Holman	1·23	1·52	43·50	76·289	76·650	32·17	16·25	$1\frac{3}{4}$, $1\frac{5}{8}$, $1\frac{1}{2}$, $1\frac{1}{8}$

everywhere and which evidently discouraged the agents themselves, is largely explained when it is seen that it was required to excavate 69·6 cub. in. or 17·6 cub. in. more than the Gordon to gain 3·7 in. extra depth of hole.

The Little Holman machine was supposed to be working stiffly and to be striking blows of little power, when in reality it was being called upon to excavate 24 cub. in. more than the Gordon to drill a hole only 3 in. longer. Turning to the performance of the Gordon drill, it will be seen that it managed to get its holes in 40·7 in. with only 52·0 cub. in. of excavation. Even the Chersen drill had to excavate 6 cub. in. extra to gain 2·3 in. greater depth of holes The Gordon engineers managed this by drilling—not the 36 in. of their hole with bits 1¼ in. and over, but only 25·41 in. The Ingersoll machine, as might be expected, also cut things pretty fine ; but it drilled 29·4 in. of its holes with bits 1¼ in. and over. Or to put it another way, the Gordon drill excavated 29·3% of the cubic contents of their wet holes bored with a 1⅛ in. bit. Personally I was under the impression that one of the advantages of the hammer drill was its ability to drill holes with a smaller difference in diameter of following bits than piston drills, but the bits used by the Ingersoll, Chersen and Kid machines were of the same size. The great advantage gained by the Gordon engineers by putting in a hole a few inches shorter than the average of the others, and by drilling that hole only some 6 in. with the starter, 8·8 in., with the second, 10·5 in., with the third and no less than 15·35 in. with an 1⅛ in. bit is now apparent, but the hole at the bottom was certainly no bigger than a hand hole, and the powder was not "where it belonged" any more than it would have been with a hand hole. I measured the bits being used in actual work by the Gordon machine on one of the mines employing them. I found that they averaged about ⅛ in. larger diameter all through than those used in the test. This, in my opinion, alone accounts largely for the very great difference of footage bored per shift in actual practice compared with that shown in the tests.

Let us try and take up the position of a manager who wishes to compare the machines with the idea of using one of them for stoping with down wet holes. He will first eliminate the comparisons of boring speed shown in the tables, as they include flat holes, and he would draw out a table of drilling speeds similar to the above shown, and read it in the light of the facts already considered. It will be seen that the Chersen, Ingersoll and Flottman show a better average than the Gordon, and that the Kid 2 in. diameter comes very close to it, despite bad steel having spoilt its performance. This tends to show that the Holman machine, and perhaps the Little Wonder, both 2 in. diameter drills, could probably drill about as fast as the Gordon, using similar sized bits. I may say that this analysis of figures surprised me, as I considered that the actual boring speed of the Gordon drill had been shown to equal a 2¼ in. machine. This result, however, tends to bear out American experience with a different type of hammer drill and 2¼ in. machine mentioned in my paper. Where the Gordon or other water feed hammer drill scores and shows its superiority is first in the ease and rapidity of erection, and secondly, in its power to drill with equal speed in any direction, but results in actual work show that if the hole is to be finished with a 1¼ in. bit the Gordon drill will require to be made heavier to strike a more powerful blow than at present. In my paper I stated, however, that I did not think that under conditions here this advantage of greater ratio of drilling time to total working time for hammer and piston drills would be so marked. It is interesting to note what this amounted to, in the case of the Gordon drill and piston drills on the surface. The margin is about 9% greater than the best handled piston drills, and would undoubtedly be greater in very narrow stopes underground, as shown by the table given by Prof. Yates, but this record in time was unfortunately not kept in the underground tests. The mine manager would, I think, be guided in the adoption of any machine from those competing by his own estimate of what ratio maintenance costs, air consumption and labour would in his particular circumstances bear to actual speed of drilling wet holes in total working time, and he would also expect all the makers to improve their machines as a result of the information gained. Prof. Yates was struck by the ease of manipulation with the wedge clamps adopted by the Gordon Drill Co., and he sees no reason why a drill like the Flottman should not be similarly equipped. I can assure Prof. Yates that an inspection of the patent records will show that this company has endeavoured to supply most excellent reasons to the contrary.

Regarding the dust question, I was sorry to see that apparently the impression has got about that small stope drills are phthisis producers, and that the advent of a small drill like the Gordon is going to reduce miners' phthisis. All the phthisis small drills have induced would do nobody much harm.

Even Lord Selborne did not seem aware that it is the larger machines used in development that are the manslayers, and that long before the Gordon drill was thought of the Leyner drill, of similar design on a larger scale, was tried here in their place and for various reasons discarded.

Turning now to the question of the relative advantages of small stope drills and hand labour, I have first to note Mr. Coombe's criticisms. Anything Mr. Coombe has to say on practical mining must be worth listening to, and one hesitates to differ from him. Mr. Coombe has no use for small drills of any description. According to him, anything they can do can be better done by hand drilling, and labour for such work will always be available. Now I do not think anyone will dispute the fact, that under favourable circumstances (which include ground not too hard or tight and stopes at such an angle that the hand hole can be drilled as near the vertical as possible, and given a miner who knows his work, and drillers who can always be depended to sink their hole to the required depth with a fair percentage of workers putting in their two holes), machine drilling in a stope of 36 in. or under must reach a much higher level of efficiency than any before attained, in order to compete successfully with hand labour. We must, however, remember that the hand driller requires one to two months at least to become efficient, thus reducing his average efficiency over his term of contract, that the ideal gang of workers is not always procurable and that a good hand stoper is as hard or harder to obtain than a good machine man. My experience has not been as varied as I could wish, but the impression I have from inspecting numerous stopes with various workers is that on the average hammer drilling and breaking of ground by 36 in. holes is not an efficient means of breaking ground in narrow and tight stopes. It would be interesting to have the average length of stumps of holes formed in a big mine every morning for a month compared with average depth drilled. As soon as the ground becomes tight and free from heads that can be drilled to, the efficiency of hand labour tends to fall off, this will be especially noticeable in the great heat of the deepest mines, and when the stopes become flat 15° to 7°, as in the far East Rand basin the efficiency will be found to be seriously reduced, for experiments have shown that while 1 in. can be drilled in granite in 188 seconds with hole looking 60° down. It takes 282 seconds to drill the same distance with hole inclined at 27° down, and stoping widths must be kept wide enough for a native to sit up and drill.

Here is the field for efficient machine stoping with small machines: small, because air consumption is a serious item, and because they can be more easily handled.

Mr. Coombe is not aware of developments of late years if he thinks that managers in other countries, where labour is expensive, use hand labour instead of small machines. The modern manager elsewhere uses hammer drills and sometimes 2¼ in. machines in his stopes. In Cornwall hand labour is cheap, for wages are low. To say that miners prefer hand labour to machines in small stopes because there is less worry and running about is only to tell us what we all know that no miner will work hard if he can do otherwise. The theory of blasting tells us that for breaking rock having only two free faces by a charge distributed along the hole, the only right direction for a hole is (as I found myself from actual experience) parallel to the free face. Such a parallel hole can be drilled best by a machine clamped in position. If the charge is properly distributed along the hole with means to secure its complete detonation long holes can be used in stopes with small individual charges, and the rock can be broken with the greatest economy. If we have not efficient workmen for this they must be trained.

Four holes thus drilled on one bench can be made to break as much or more ground than four similar average hand holes on separate benches. There is, however, no reason why a light hammer drill with air feed backed against a temporary spray or bar should not be used to put in one hole on one bench if required. As Mr. Lane Carter has pointed out the requirements of different parts of the field vary greatly. That machine that would suit the soft ground on the West Rand will not suit hard banket in the deeper mines. There is room, as I pointed out, for a variety of types, and despite the reported disappointing results of the Gordon machine on certain mines, due to the causes I have mentioned, I think it, or machines like it, will have its place here. Wastage of water is, however, a serious item, as the mines are not too careful to supply the water at just the right pressure to inject water and yet to avoid excessive leakage or to supply water free from grit and mud.

Drill Bits and Cutting Edges.—Mr. Anderson has, I think, shown us as I contended that we have much to learn. If his bit, as may be hoped, fulfils expectations working on a large scale it will be the finest advance made on Rand mining methods for very many years past. The causes of failure of detachable bits in the past are well known, and Mr. Anderson may have succeeded in overcoming them. His experiments have, as shown by Mr. Roche, already borne fruit in the improvement of ordinary bits.

Konomax Drill.—I am informed by a manager using a 3¼ in. machine that it does the same work per shift as a 3¼ in. Ingersoll with 60 lb. pressure, and uses less air. The questions of wear and leakage are not however solved. The Little Konomax is still in the experimental stage. Hitherto all attempts to pass water down the piston and chuck into the shank of the drill have proved failures, owing to the leakage set up by wear and tear and the water getting into cylinder and valve parts of machine. This trouble is by no means absent in the Gordon and other hammer drills.

I do not think I or anyone else have reliable data to criticise Prof. Yates' figures regarding the number of hammer boys that can be replaced by a perfect stope drill. It will depend so much on what the drill can do.

I should like, before closing, to again consider the question of hammer and piston drills for stoping in the light of the tests and other later evidence. Generally these trials are held to have shown that the hammer drill is a more efficient machine than the piston drill. In other words, a larger percentage of the total Kinetic energy of the blow is spent in useful work in cutting rock. This may be true, but in a hammer drill quite a large percentage of the force of blow is absorbed overcoming the inertia and elasticity of the drill steel and anvil block, and the longer the steel the greater the loss, this is perhaps well shown in the above table. The increase of drilling speed of the Gordon drill with 1⅛ in. bits is not as marked as with piston drills owing to this loss being greater with the long steel. What these losses are with hammer and piston drills is hard to determine experimentally. The force of the blow in foot pound struck on lead can be determined by comparing the indentation made with that caused by a weight falling from a given height: but this gives no idea of the cutting efficiency in actual work. The causes of loss with the piston drill are several.

There is friction in the machine itself. There is friction of the drill bit on the sides of the hole caused, firstly, by vibration on the supports of the machine; secondly, by any variation between the axis of the drill bit and the true axis of the machine caused by the chuck arrangements being ill designed or worn, or by the drill bit being bent or badly sharpened. There is loss due to lack of rigidity in the striking tool, as shown by Mr. Anderson. There is loss of power, caused by doing work twice over in pulverising rock already broken from the solid at the bottom of the hole. There is the loss due to excessive blunting of bits due to their loss of temper by heating in certain holes.

There has been the difficulty met with in using air expansively in piston drills, whereas these difficulties are not so apparent in hammer drills. There is inefficiency caused by variations of the stroke. Now it cannot be said that the hammer drill as exemplified in the best type seen in the trials has much "up its sleeve."

Further use of air expansion might perhaps cause some little economy in air consumption, but would not probably be worth troubling over. The cuttings are ejected by water as soon as broken. The bit is kept cool. Friction on the sides is reduced as the bit is not withdrawn at each blow struck. There is indeed a better chance of applying electric power, but the Box drill constructed on this principle has apparently been a failure in hard rock.

We have now to ask what has the piston drill "up its sleeve?" in other words, what has been done, what is being done, and what can be done to improve its efficiency and reduce these losses? Friction in the machine can be reduced by some scheme of automatic lubrication. This is quite possible. Loss by vibration of the mounting must always exist, it can be reduced by shortening the stroke and by increasing the number of blows per minute. The facts brought forward by Mr. Anderson, as already pointed out, may go very far to eliminate these losses and greatly increase boring efficiency. Combine this with hollow steel with air passing down it, or air and water, or water alone, and again we have a great step towards economy of power. This is being attempted by myself and in the Little Konomax and the Climax machines. The Konomax machine has undoubtedly showed the possibility under certain circumstances of reducing air consumption. The Little Konomax machine shows the possibility of making a piston drill as handy as the Gordon drill, having also an automatic feed to keep the stroke at the maximum. The use of higher air pressures enabling a smaller machine to do the work at present done by a large machine (on a shorter stroke when hollow steel is used) will also, I am certain, effect a great revolution in rock drilling. There is, I believe, nothing in the result of these trials to discourage enterprise and invention in the improvement of design of small piston drills.

So many engineers seem still doubtful of the advantages of using air at 80 to 90 lb. with machines that I ask leave to be forgiven for again harping on this point. I would say first that this question has been settled already outside the Rand, and on all progressive mining fields it is recognised that rock drill machines should be supplied with air at 80 lb. or over at the machine. It has also been recently shown that given good material and workmanship in laying and maintaining pipe lines, air is transmitted more cheaply at high pressures than at low ones. It may also be taken as proved that, other things being equal, speed of boring is proportional to the number of blows struck in a given time. In the trials the increased speed of drilling shown with a rise of only 10 lb. pressure was 27% on an average for all the machines. Some time ago in California Messrs. Halloran & Hamilton conducted the only exhaustive and reliable series of tests I know of on a Corliss valve tappet drill. They showed that with a certain length stroke, which may be different with different machines, the rate of speed of reciprocation becomes directly proportional to rise of pressure, one engineer states it is proportional to the square root of pressares used, being in the case of machine using 3½ in. stroke a rise of 95 strokes per minute for rise of pressure of 10 lb. Or from 80 to 90 lb. pressure the rate of reciprocation rose from 835 strokes to 920 strokes per minute. Prof. Yates and other engineers are, however, frightened that wear and tear may be excessive and grope for an unhappy mean between 60 and 70 lb. Here I think we need to grasp clearly what this contention means. What do Prof. Yates and others mean by extra wear and tear caused by high pressures?

This extra wear and tear may be due to two causes. Firstly, to extra speed of movement. The piston cylinders and valves being subjected to more movement and wear in a given time need renewing in a shorter time than they would with low air pressures. Let us express the life of a machine in holes bored. Then the science of mining finance must teach us that it will pay to have a machine with a life of only six months if in that time it bores the same number of holes bored by another machine in 18 months. Surely it is better to wear out a machine than to rust it out. Hence on this ground no exception can be taken to working with high pressures. Wear and tear may, however, be caused by extra jar and stresses set up on machine and steel by the more powerful blow struck. If the blow at present struck by a 2 in. or 3¼ in. machine with 6 in. stroke and air at 60 lb. is powerful enough to chip and crush the rock—well and good—the only trouble is that the number of blows struck per minute is too few. With 80 or 90 lb. pressure with the same machine the force of blow may or may not exceed the economic limit. If it does not exceed this limit, with the increase in number of blows per minute we get greatly increased speed of boring. If it does exceed the economic limit we can either reduce the size of machine cylinder or the length of stroke and get increased boring speed with the same expenditure of air as with lower pressures.

I think anyone will find a 2¾ in. machine at 80 lb. will beat a 3¼ in. machine at 50 lb. The best economy will, however, I believe, in the end be found by shortening the stroke and by using hollow steel with air or water injection, with a cylinder diameter made of such a size as to strike a just powerful enough blow with air at 80 or 90 lb. Personally I believe in boring rock with a large number of light flows per minute delivered by a sharp tool. Such a machine should suffer no more from jar and undue stresses than any present machine. This argument does not mean that it would pay to try and force air at 90 lb. through the miles of air pipe in our older mines; but it is a plea, supported I think by the strongest evidence, that it will pay to install compressors to deliver air at 100 lb. in our new mines. I make bold to say that given air at the faces of our mines at only 75 lb. the stope drill problem would not exist even with the present types of small machines available.

I must apologise to the agent of the Climax machine for stating that no air economy was practised by allowing for expansion. If it has been found practicable to cut off at half stroke an advance has been made. It will be interesting if Prof. Orr can find time to compare the relative air consumptions of this machine and the Konomax. If I have failed to notice any other criticism I must ask pardon, as I am afraid my reply has already exceeded the economic limit in length.

The President: Many of us came to-night fully expecting to hear something interesting about rock drills, and we have not been disappointed. The paper itself was really excellent, and I am sure I am only echoing your own views when I say that the reply is equally so. In my opinion, the paper and the reply constitute one of the finest contributions on rock drills we have ever had. I have much pleasure in again thanking Mr. Weston.

WESTRALIAN WET-CRUSHING PLANTS, WITH SOME NOTES ON LABOUR EFFICIENCY.

(*Read at February Meeting, 1908.*)

By Gerard W. Williams, F.C.S., A.I.M.M.

DISCUSSION.

Mr. H. A. White: I have to congratulate Mr. Williams on his very ably written paper, and to thank him for the most complete and systematic account of metallurgical practice in Western Australia which we have had before our Society. That part of the subject which deals with labour efficiency comes at a very opportune moment, as it deals in some measure with a question investigated here by a Government Commission, whose report is now before the public. Incidentally, I may remark it lends very meagre support to the idea that all white labour could be used successfully on these fields for the costs quoted for treatment on the Golden Horseshoe, by a process sought to be introduced here on the score of economy, are 12s. 10d. against our local figure of about 5s. If mining were to be increased in cost on the same scale we might as well close the mines without indirect methods and need not even consider that the rates of pay for white men are actually lower in Australia than on these fields, and thus accentuate the difficulties involved.

I venture to think that there is sufficient truth in Mr. Williams' accusations with reference to waste of native labour to make it worth while for every battery and cyanide manager on these fields to go very carefully into the possibility of economy in this direction, and I acknowledge I have benefited by his suggestions. Of course, I do not mean that big reductions can be made, but as I have elsewhere remarked, we have now come to the days of small things, which are nevertheless not to be despised.

Going into detail, I am surprised by the statement that on the sands plant pumping is apparently done on one shift only, as only one man is charged for the three shifts, and I consider that efficiency must be sacrificed. The provision of 28 men for discharging and closing filter presses points to the inadequacy of mechanical appliances.

With reference to Table "A," it would be interesting to know the reason of the very high relative duty of the mill at the "Sons of Gwalia," and to be certain that 108 drops a minute of 8 in. are actually obtained at that mine. It does seem a great pity that the obsolescent practice of adding mercury inside the boxes is still in use in a country where so much outcry is made about gold thefts, and makes one somewhat sceptical of the claims of advance in metallurgy so freely made there. The toy tube mills of 3 ft. 3 in. diameter by 13 ft. should be donated to some mining college for experimental purposes, and we should hear less of such costs for tube milling as 2s. 6d. a ton and such infinitesimal outputs as 27 tons per diem. No comparison with grinding pans of such miniature apparatus can be of any real service, and if the pulp contains as much as 75% of water on entering the tube mills an opportunity of ascertaining their value under proper working conditions is lost.

With reference to Table "D," it is impossible to refrain from expressing astonishment at the exceedingly low extraction, which can only be accounted for on the assumption that the problem of dealing with 13 dwt. ore containing tellurides is still open to a profitable attack. The figures do not compare at all favourably with up-to-date Rand practice, and taken in conjunction with working costs make one wonder quite what the author means by "applied with equal success under alien conditions." I do not mean to accuse Mr. Williams of proposing these methods for our adoption, but merely wish to point out that if only equal success attended them here, every company that adopted such a "new metallurgy" would incur a heavy loss as compared with our present practice yielding up to 95% extraction for a reduction cost in the neighbourhood of 5s.

NOTE ON THE DAILY VARIATION OF RAND MINE VENTILATION.

(Read at November Meeting, 1907.)

By James Moir, D.Sc., M.A., F.C.S. (Vice-President).

REPLY TO DISCUSSION.

Dr. J. Moir: Though there has been no discussion on my paper, I wish to amplify the point raised in its last paragraph regarding probable increase of ventilation-pressure at greater depths. To do so, I require to improve on the usual formulæ so as to deal with most of the factors concerned :—

(1) *Effect of Inequality of Level of Shafts.*—Let H be depth of upcast from collar to middle of connected workings ; let h be the difference of level at the collars, so that $H+h$ is depth of downcast if it be the higher, or $H-h$ if it be the lower of the two. Let T_0 be the air temperature, T the average temperature of upcast, and t the difference between the average temperatures of the two shafts. The ventilating-head is the difference of the heads of the two columns :—

$$\therefore X = \underbrace{\frac{(H+h)\,(273+T_0)}{273+T-t}}_{\text{Downcast}} - \underbrace{\left(h + \frac{H(273+T_0}{273+T}\right)}_{\text{Upcast}}$$

This simplifies to :—

$$H.\frac{273+T_0}{273+T}.\frac{t}{273+T-t} \pm h\frac{T_0-T+t}{273+T-t}$$

Now the factor $\frac{T_0-T+t}{273+T-t}$ is always less than ·04, and in moist weather is zero, consequently the effect of the ordinary inequalities in the levels of the shaft collars is almost negligible. Again, for all ordinary purposes the air and upcast temperatures can be assumed equal (within a few per cent.), and the practical formula for ventilating head thus becomes $X = H\frac{t}{273+T} \pm \frac{h}{40}$: Thus if $H = 1{,}500$ ft., $h = 15$ ft., $t = 10°$ C. $= 18°$ F., and $T = 20°$ C. $= 68°$ F., we find a ventilating head of 51·6 ft. when the upcast is the lower, and 50·8 ft. when the upcast is the higher. For rough calculations one can assume a constant value for T, and the formula is then $X = H\frac{t}{295}$ within 3%. In Fahrenheit work this becomes

$X = \frac{Ht'}{530}$ where t' is the difference of temperature of the shafts in Fahrenheit degrees.

(2) *Ventilation Pressure.*—This is head multiplied by density.[*] The density of dry air is 0·0807 lb. per cub. ft. at N.T.P., hence generally $D = \cdot 0807 \times \frac{491}{459+T} \times \frac{B-f}{29 \cdot 92} = 1 \cdot 325 \frac{B-f}{459+T}$ lb. 1 cub. ft., where B = barometer and f = actual vapour pressure of water, both in inches. $(B-f)$ does not vary much and may be assumed = 25·0 in shallow mines, when measured half way (viz., 800 ft.) down for an average: hence in shallow mines $D = \frac{1 \cdot 325 \times 25}{527}$ or 0·0629 lb. per cub. ft. Even in the Jupiter mine (4,100 ft. depth), $B-f = 25 \cdot 8$ on the average, hence since T = 75, D = ·0641. Hence in the above imaginary case of 18° F. difference between the shafts, the ventilating-pressure would be 0·0629 × 51·2 or 3·22 lb. per sq. ft. or 15·7 mm. on the water gauge.

(3) *Increase of Ventilating Head with Depth.*—To estimate this we have to find the relationship on a deep mine of t' in the formula to the depth under similar conditions to those indicated above. In the first place the average temperature of the upcast depends on three factors (1) increased rock temperature, at the rate of 1° F. per 210 ft., (2) increased adiabatic decompression, and (3) falling of cooler water from near the surface. According to factor (1), one would expect the temperature at the bottom of a 4,000 ft. shaft to be 82° F. here, and according to (2), if it rose to the surface quickly it would cool to 62° according to Kirkby's formula $\left((t+34)\left(\frac{760}{B} - \cdot 31\right) = K\right)$:[†] but as a matter of fact the air takes about 15 minutes to ascend and is much affected by the falling water, so that it does not cool anything like so much as 20°. In this connection I wish to quote some very interesting readings obtained by Mr. McArthur Johnston on the Jupiter mine and kindly put at my disposal by the Consolidated Goldfields. The air temperature at the foot of the upcast (4,100 ft. from surface) was about 80° F. in a fair current, so that the rock temperature is probably about 82° in reality, just as calculated above: at the 3,400 ft. level, below 1st drive, it was 78·5, and at the 3,350 ft. level it was 77·0 on the average, whilst near the surface the average was 69·5, a fall of only 7½° in making a clear ascent of 3,300 ft. This shows clearly the marked effect of the falling water in preventing the air from cooling to its full extent. The figures given are the means of maximum and minimum thermometer readings at the places mentioned, the instruments having been left *in situ* over a week-end. It is probable therefore that the true average temperature of the Jupiter upcast[*] is very close to 75° F., whereas that of the Ferreira Deep (assuming 1,600 ft. depth) is 68° F.

I have made observations on this point of average upcast temperature on some other mines. (*a*) The Robinson G. M. gives 70° F. at the 15th level (1,089 ft. vertical), and has probably an average of 67° F. on the whole upcast. The highest temperature observed in close places of this mine was only 71½° F. on the same level. At the time the foot of the downcast read 68½° F., and the humidity was 97% or more all over the mine. The surface wet-bulb read 62° (which is unusually high) and the current was, naturally, weak, being about 20,000 cub. ft. per minute. (*b*) The Village Deep gives 75½° F. at the foot of the upcast (2,130 ft. vertically), and the average is probably about 71° F. The highest temperature in close places was 77° F. The foot of the downcast read (2,300 ft. down) 71° F. with 92% humidity, the surface temperature being the same with 48% humidity; near the top, in the falling water, the temperature was about 65° only. The air is evidently first cooled by evaporation and then warmed up by compression.

(*c*) The Robinson Deep (observations by Dr. Irvine) gave 77° F. at the bottom of the upcast (2,660 ft. vertical), and its average upcast temperature is about 72°: 77° was also the highest temperature observed in close places: the foot of the downcast (2,500 ft. level) was at 70½ (saturated) when the entering air had a temperature of 75° and only 50% humidity. The air supply was poor.

Arguing from these results we see that increased depth certainly does increase the average temperature of the upcast, but the increase is comparatively small and seems to vary as the square root of the depth, the equation being something like $T = 60 + 0 \cdot 245 \sqrt{H}$.

To estimate t' in the formula it is necessary next to find a mathematical expression for the average downcast temperature of deep mines—a problem in itself extremely difficult with the present data. In shallow mines I assumed in the original paper as the result of experiment that this quantity might be put equal to the surface wet-bulb temperature, because in such mines the cooling effect of falling water on dry air is by far the predominant factor. As a makeshift I shall assume in the same way that in deep mines the downcast air first falls to the temperature of the wet-bulb in the first thousand feet and then slowly warms up owing to compression; but as the effect of compression is lessened by the falling water I shall instead of using the usual exponential formula for compression, simply suppose the rise of temperature proportional to the depth *i.e.*, assume $t_1 = w + \frac{H - 1,000}{c}$ where w is the surface wet-bulb and c is a constant which, from the observations, is, roughly, 250.

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[*] See also T. Johnson, this *Journal*, vol. iv., p. 239.

[†] See Dr Haldane's Blue-Book "Health of Cornish Miners," appendix iii., p. 97.

[*] The temperature of the downcast was 70° (with 53% humidity) at surface, and 70½ (saturated) at the bottom. It is, however, much cooler in the intermediate portion.

Now, since we found $P = \frac{0·063}{530} Ht'$, therefore, by substitution we find

$$P = 0·000119 H\left(60 + 0·245 \sqrt{H} - w - \frac{H - 1000}{250}\right)$$

$$\text{or } P = 0·000119 H\left(64 - w + \left(\frac{\sqrt{H}}{4·1} - \frac{H}{250}\right)\right)$$

approximately. Now the factor $\left(\frac{\sqrt{H}}{4·1} - \frac{H}{250}\right)$ is approximately four at medium depths, vanishes at 5,000 ft. and becomes negative afterwards: for our purpose it can therefore be neglected. In the meantime it may be noted from the formula that the ventilation-pressure undoubtedly increases with depth and practically in direct proportion thereto.

Ventilating Current.—This depends on ventilation-pressure, shaft section and the total friction of the mine. I adopt the convenient form of the usual formula given by Thos. Johnson* (the friction constant of which I have verified by experiment), namely, $Q = A\sqrt{\frac{PA}{KS}}$ where $K = \frac{1}{200}$ A is 80% of the shaft area between timbers (omitting ladderways, etc.), and S is the total rubbing surface of shafts and workings in square feet.

Now, in most of our mines the shafts have three hauling compartments, and their area is about 75 sq. ft., hence A is practically 60. As regards the frictional surface S, it is composed of that of shafts and workings. The shaft portion is twice the perimeter into the depth (or 80H) whilst the workings, counting only main connections, amount to $n\,l\,p$, where n is the number of main connections, and l and p their length and perimeter. To discuss the effect of increased depth on quantity of air assume two mines with four through levels, but one shallow and the other deep, the depths being H and H' respectively. In the first S will equal ($80H + 4 \times 2,000 \times 20$). In the deeper mine S equals ($80H' + 160,000$), the shafts being assumed 2,000 ft. apart. Hence, going back to the quantity formula, the ratio of the air quantity in these two mines, namely, $\frac{Q'}{Q}$ equals $\sqrt{\frac{P'}{P} \times \frac{S}{S'}}$ and since $\frac{P'}{P} = \frac{H'(64 - w)}{H(68 - w)}$ from the considerations mentioned on the previous page, we find finally that

$$\frac{Q'}{Q} = \sqrt{\frac{H'(64 - w)\,(H + 2,000)}{H\,(68 - w)\,(H' + 2,000)}}.$$

For a numerical comparison let H = 1,500 ft., H' = 4,500 ft., and $w = 56°$ F.; then $\frac{Q'}{Q} = \sqrt{3 \times \frac{8}{12} \times \frac{3,500}{6,500}} = 1·04$, *i e.*, the increase of air quantity with depth will be extremely small in summer. In cold weather, however, the wet-bulb is much lower, and when it registers 41° F., for example, the value of $\frac{Q'}{Q}$ will be 1·175 indicating an increase of 17% in air supply at depth.

I regret the unsatisfactory and empirical manner in which I have been compelled to treat this exceedingly complicated subject: the only consolation I can give in conclusion is that there seems to be no reason whatever to fear a diminution of total air supply as the mines become deeper.

Contributions and Correspondence.

NOTE ON THE CAUSE OF CERTAIN MISLEADING ANALYSES OF AIR.

On a hot day when the laboratory temperature is, say, 72° F., and the vapour pressure of water is consequently as high as 20 mm of mercury, a serious error may result in the readings of gas-volumetric apparatus, and I find that my colleagues do not allow for this source of error. 20 mm of vapour pressure corresponds at this altitude to about 3·23% of water vapour in the air; consequently in analysing air over water the oxygen percentage of the dry air is apparently depressed by about 0·7%; whilst the difference (reported as nitrogen) is affected by two errors, viz., the above 0·7% and in addition the actual quantity of water vapour present. Thus a sample of (fairly fresh) air containing when dried in reality 20·95% oxygen, 0·05% CO_2 and 79·0% of nitrogen (+ inert gases), when introduced into an Orsat (for example) at 72° F., will expand by 3·33%: and the analysis will give $CO_2 = 0·05\%$, $O_2 = 20·25\%$ residue = 79·7%. The man in the street, seeing such an analysis, would suppose that some oxygen had been used up in such air—which is by no means the case, the apparent deficit being entirely due to dilution with water-vapour. Again, of the residue 3·2% is water vapour, so that the true value of the nitrogen (+ inert gases) in the saturated sample is 76·5%: the depression from 79·0% corresponding exactly with the dilution of the dry gas with 3·33% of water vapour.

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* See this *Journal*, vol. iv. (1903), p. 241.

Consequently, to get comparable results with such apparatus, it is necessary to recalculate the observed readings. It would be more correct theoretically to calculate the results back to the condition of the air as sampled, *i.e.*, to use in the calculation the actual percentage of water vapour present at the time of sampling (which is: maximum vapour tension × percentage humidity ÷ 100); but in practice it is usual to calculate back to dry air, so as to be able to compare directly with the existing standard analyses of fresh air. Thus if the observed readings are: $CO_2 = 0·1$, $O_2 = 20·2$, residue = 79·7, and the temperature of the water is 70° F., the barometer reading 620 mm., *i.e.*, vapour pressure = 18·6 mm., then percentage of water vapour = $\frac{18·6}{620} \times 100 = 3·0$ %. Consequently 100 c.c. of the wet gas contains $CO_2 = 0·1$ c.c., $O_2 = 20·2$ c.c., N_2, &c. = 76·7 c.c., $H_2O = 3·0$ c.c.; which quantities are also contained in 97 c.c. of the dry gas; therefore the true analysis of the dry gas is (multiplying the figures by $\frac{100}{97}$). $CO_2 = 0·103$ %, $O_2 = 20·825$ %, and N_2 &c. = 79·072

An example of such a fallacy is the deficiency of oxygen always observed in mine air. All mine airs are nearly saturated, and consequently dry air entering the downcast with, say, 21 % oxygen will through mere dilution with water-vapour contain, say, 20·5 % at the foot of the shaft. In spite of this elementary fact, many scientists have attributed the fall in oxygen to such things as oxidation of pyrites or timber.

It should also be noted that the state of humidity of the air also affects the results of CO_2-determinations by Pettenkofer's method, but only to a slight extent. The observed barometric pressure should be diminished by the existing vapour tension at the time of sampling.

JAMES MOIR.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

ACTION OF ALKALINE SOLUTIONS IN CYANIDING.—"It is a well-known fact that the presence of sulphates of iron in tailings, resulting from the decomposition of their pyritic constituents, has a very harmful effect on cyaniding. In every case where partially weathered sulphide tailings are treated the consumption of cyanide is above the normal, to a greater or less extent, according to the amount of decomposition products present, and in order to reduce the consumption the practice of using preliminary alkaline and water washes has been in many cases successfully adopted. It is in connection with the use of alkaline washes—or, more particularly, with the use of protectively alkaline solutions—that these notes will deal. The material experimented on was the accumulated sands at the Burbanks Birthday G. M., and the following are a few results.

An estimation of the sulphur contents yielded:—

S as sulphates	0·125
S as sulphides .	1·52

The sulphates are not all iron salts, but include $CaSO_4$ and $MgSO_4$ absorbed as 'moisture' from the heavily mineralised water in which the ore was crushed.

The cyaniding of this sand with a solution containing KCN only, involves a heavy consumption of this chemical.

Table I. gives the results obtained by bottle (agitation) tests on this material, using varying quantities of NaOH and showing quantities of cyanide consumed.

Table I.—Equal Weight of Sand and Solution.

	NaOH added.	KCN.	KCN Consumption per ton.	Time Treated	Before Treatment.	After Treatment.	Extraction.
	Percent	Percent	lb.	Hours.	s.	s.	Percent
A	Nil	0·2	2·2	48	14	3	78·6
B	0·2	0·2	0·4	48	14	11	21·4
C	0·4	0·2	0·4	48	14	11	21·4
D	0·8	0·2	0·4	48	14	11	21·4

The consumption of alkali by the ore is 0·15 %, given in terms of NaOH, so that B does not show very great excess in alkalinity. In B, C and D the solutions after the treatment gave protective alkalinity, and the figures in the third column give the cyanide consumed in pounds per ton (a saving of 1·8 lb. in the alkaline solution). On the other hand, the final column shows an enormous drop in the extraction with the solutions containing NaOH. Further, this drop is apparently not influenced by the amount of NaOH present, provided it is in excess of the quantity required by the ore. By using a smaller quantity of NaOH than that required to give protective alkalinity, it is possible to get a higher extraction, but, of course, by doing this the chief function of the alkali—the saving of cyanide—is impaired.

In seeking a reason for this loss in efficiency it seems likely from the nature of the ore that the formation of soluble sulphides would account for it, but, on the contrary, in the solutions from all the tests using excess of NaOH not even a trace of those compounds could be detected, although a considerable amount of sulphur was combined as thiocyanates.

Now, as the presence of NaOH in the solution causes loss of dissolving efficiency, the latter is, in all probability, due to the action of that reagent on some constituent of the sand other than iron sulphide, for instance, ferrous sulphate ($FeSO_4$), which is known to be present in appreciable quantities. With ferrous sulphate NaOH would tend to form ferrous hydroxide ($Fe(OH)_2$), and this being a very powerful reducing agent might easily explain the reduced efficiency. If this assumption be correct, then the preliminary removal or oxidation of the $FeSO_4$ should improve the extraction obtained with an alkaline solution. A sample of the sand was agitated with dilute sulphuric acid and then filtered and well washed with water until only a trace of acid showed in the washings. This was then agitated for 48 hours with an equal weight of solution containing 0·2 % KCN and 1 % NaOH, exactly the same treatment as the tests in Table I. received. Before treatment, 14s.; after treatment, 4s. 6d.; extraction, 67·8 %. The extraction did not reach the maximum, but this is probably explained by the incomplete removal of the $FeSO_4$.

Another sample was thoroughly dried and turned over, practically all the ferrous sulphate being oxidised by this means to the ferric state, and consequently would form the harmless ferric hydrates with NaOH.

This material under the same test gave as follows : Before treatment, 20s. ; after treatment, 5s.; extraction, 75% ; which nearly reaches the maximum.

Coincident with this last test another was made with the same dried material and under precisely the same conditions except that 0·2% of iron ammonium sulphate was added [0·2% of the salt $FeSO_4(NH_4)_2SO_4.6H_2O$ is equivalent to 00·8% of $FeSO_4$]. Before treatment, 20s ; after treatment, 17s.; extraction, 15%. The fact that 0·08% of $FeSO_4$ is sufficient to lower the extraction to 15%, or a difference of 60% in the amount of gold dissolved is quite sufficient to demonstrate its harmfulness.

A confirmatory test was also made on an oxidised ore, entirely free of sulphides and the extraction was not adversely affected by excess of NaOH :—

Table II.

	$FeSO_4$.	NaOH.	KCN.	Before Treatment.		After Treatment.		Extraction.
	Percent	Percent		s.	d.	s.	d.	Per cent.
A	—	1	0·2	7	0	1	6	78·6
B	0·08	1	0·2	7	0	3	0	57·1
C	0·32	3	0·4	7	0	6	6	7·0

The effect on this ore, though not so marked, is nevertheless very noticeable.

In the solutions from all these tests thiocyanates are present, and quantities of ferrocyanides are found in the solutions from the non-alkaline treatment. In the alkaline solutions, however, the merest traces, or in most cases no ferrocyanides at all can be detected. This fact supports the following explanation :—

In non-alkaline solution $FeSO_4$ and KCN would react : $FeSO_4 + 6KCN = K_4FeC_6N_6 + K_2SO_4$ with other similar reactions all tending to form ferro- and perhaps ferri-cyanide compounds which would not be in the main oxygen-absorbing.

With alkaline solutions, on the other hand, a great part, if not all of the $FeSO_4$ would react with NaOH, $FeSO_4 + 2NaOH = Fe(OH)_2 + Na_2SO_4$ forming $Fe(OH)_2$, a very powerful absorbent of free oxygen. An action such as this would account for the inferior extraction in alkaline solution owing to the absence of free oxygen. Of course, the $Fe(OH)_2$ takes time to bring about the reduction during which more or less of the gold would be dissolved, and this is a good explanation of the less marked effect of the $FeSO_4$ on the oxidised ore in which the gold is more soluble than in sulphide sands. Similarly, the better results obtained with an insufficient quantity of alkali can be explained by the fact of a smaller quantity of $Fe(OH)_2$ being formed, and consequently a slower absorption of oxygen.

The conclusion which must be arrived at from these experiments, is in favour of the non-alkaline solution with perhaps a preliminary water wash, in spite of the heavy consumption of cyanide. On the above mine the consumption of cyanide has been reduced to 1·6 lb. per ton for an average extraction of 79% under favourable circumstances, *i.e.*, with fresh solutions that have not become fouled with frequent use."—BEAUCHAMP L. GARDINER.—*Monthly Journal of the Chamber of Mines of Western Australia*, Oct. 31, 1907, p. 915. (H. A. W.)

INTERNATIONAL ATOMIC WEIGHTS (1908).—

Element.	Symbol	0=16
Cerium	Ce	140·2.
*Dysprosium	Dy	162·5
†Indium	In	115·0

There is only one alteration in the atomic weight given in the *Journal* last year (see *Journal*, vol. vii. Feb., 1907, p. 270). — Proc. of Chemical Societ (London), Jan. 25, 1908. (W. A. C.)

FIFTH ANNUAL REPORT OF THE COMMITTE ON ATOMIC WEIGHT.—The number of atomic weigh determinations published during 1907 was not large but the quality of the work done was remarkabl high. The data now available are :—

1. *Hydrogen.*—Noyes made complete syntheses o water in five series of determinations, the first o which was defective, and therefore unpublished with the following results :

	ratio $H_2 : O$	ratio $H_2 : H_2$C
Second series ..	1·00787	1·00789
Third series ...	1·00767	1·00774
Fourth series ...	1·00809	1·00815
Fifth series ...	1·00786	1·00788

The most probable value from these determinations is given by Noyes as 1·00787.

2. *Silver : Nitrogen.*—Richard and Forbes have determined the ratios Ag : $AgNO_3$ which they find to be 100 : 157·479. If Ag be taken as 107·930 then N=14·037. But if N is 14·008 then Ag=107·880.

3 —*Potassium.*—Richards and Müller have determined the ratio KBr : AgBr, giving K=39·1135. And also the ratio KBr : Ag. giving K=39·1143, assuming Ag=107·93 and Br=79·953.

4. *Sulphur.* — Richards and Jones converted Ag_2SO_4 into AgCl by heating in HCl vapour. The ratio was found=91·933, which, if Ag=107·93 and Cl=45·473, gives S=32·113.

5. *Lead.*—Baxter and Wilson, in a preliminary paper, analysed lead chloride and find Pb=207·19, when Ag=107·93 and Cl=35·473. This value i much higher than the accepted value 206·9.

6. *Palladium.*—Woernte reduced palladosamin chloride to Pd, and found as a mean of seven determinations Pd=106·708.

7. *Nickel.*—Barkla and Sadler studied the absorption of secondary Röntgen radiations from variou metals, which is an atomic function, finding regularities to which Ni is an exception. If Ni has a highe atomic weight, the anomalies disappear. Te measurements for Ni as compared with other metal gave values from 61·15 to 61·6, but these must b accepted with caution.

8. *Indium.*—Mathers determined the ratios $InCl_3$ 3AgCl and $InBr_3$: 3AgBr and found 114·88 and 114·9 assuming Ag=107·93, Cl=35·473, Br=79·953.

9. *Tellurium.*—Baker and Bennett, in a researc to determine the homogenety of the element determined the percentage of oxygen in TeO_2 and the percentage of Te in $TeBr_4$ prepared by direct union i an atmosphere of N_2. The former gave Te=127·609 the latter Te=127·601, assuming Br=79·96. Marekwald prepared pure telluric acid H_6TeO_6 and reduce it to TeO_2 and found Te=126·85.

10. *Radium.*—Madame Curie has redetermine the atomic weight of radium by analysing RaCl with purer material in larger quantity than before Three determinations gave 226·18.

11. *Gaseous Densities.*—Guye, in an elaborate discussion of this subject has finally adopted the follow

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* Given for the first time. † Printed 1·15 in error last year.

ing as the weight of one litre in grams for each gas considered :

Oxygen	1·42900	Nitrous oxide	1·9777
Hydrogen	0·08987	Hydrogen chloride	1·6398
Nitrogen	1·2507	Ammonia	0·7708
Carbon monoxide	1·2504	Sulphurous oxide	2·9266
Nitric oxide	1·3402	Air	1·2928
Carbon dioxide	1·9768		

Determinations published during 1907.—F. W. Clarke.—*Journal Amer. Chem. Society*, xxx., 3, 309, March, 1908.

METALLURGY.

A Method of Leaching Gold Ore Tailings.—"Having recently (March, 1906) had to start up a plant for cyaniding old stamp mill tailings, of which 60% passed on 80 mesh sieve, the following method was found successful in accelerating leaching:

Twelve days were required at first to leach the ore; this was subsequently reduced to an average of seven days.

It was found on draining off the first solution that the charge had contracted from 6 ft. to 5 ft. in depth, which made the succeeding drainages very slow.

By slow and careful upward leaching in the first instance, conducted so as to avoid channeling, then continuously draining off from below and adding fresh solution above the ore sufficiently often to keep the surface covered, the charge sank only 3 in. instead of 12, with the above-mentioned saving in time.

Continuous upward leaching, *i.e.*, allowing the solution to overflow the vats into a launder, although it prevented contraction of the charge, would cause channeling.

Aeration of solutions was satisfactorily effected by spraying the zinc-box effluent into the sump. The method should lend itself to the handling of ore vat charges in a double-treatment plant; 80–85% extraction was continuously obtained at a cost (in August) of 1s. 10d. per ton running expenses."—R. S. Botsford.—*Western Chemist and Metallurgist*, Trans. Institute Mining and Metallurgist, Oct., 1907, p. 178. (H. A. W.)

Toughening Metals.—"In a novel method of toughening metals lately introduced the metal to be treated is placed in a closed retort with a small quantity of mercury, the retort is subjected to pressure, and as it is heated to a point below melting temperature a current of electricity is passed through the metal. Besides increased toughness, greater resistance to sea-water corrosion is imparted. The hardening is especially adapted for iron and steel, but is claimed to be useful for other metals."—*Engineer*, London.—*New Zealand Mines Record*, Nov. 16, 1907, p. 151. (A. R.)

Uses of Vanadium.—"Vanadium is a silvery white metal of high melting point—said to be 2,000° C., and, at all events, higher than the melting point of platinum—so that its commercial use in the metallic form is practically restricted to its use as a refractory metal (in the filaments of electric lamps, for instance). An alloy of iron with vanadium in the proportion of two parts of iron to one part of vanadium has a melting point of 1,375° C., or more than 100° C. below that of mild steel; and it is in this form that the metal is marketed for the use of the steel maker.

With the same ductility, an increase in the strength was looked upon as the one desirable thing, although it is now recognised that this improvement in strength, if attained at the expense of dynamic properties in the original steel, is, in most cases, of comparatively little use to the engineer.

Since, in modern machine construction, and especially in those parts which are liable to failure in use, we require above all, superior resistance to repeated stresses, to alternating stresses, to simple, repeated, or alternating impacts, and to fatigue (which is the outward and visible sign of intermolecular vibratory deterioration), a new field has been opened, and in this field vanadium has been found, by extended experiment and practical experience, to be uniquely valuable. As to static properties, vanadium intensifies greatly the strengthening effect of another ingredient, such as chromium, thus enabling the proportion of the latter to be so reduced as to avoid 'poisoning' the metal with regard to desired dynamic qualities. Moreover, vanadium is itself highly contributory to the dynamic excellence of mild steel. By retarding segregation, it renders the metal particularly susceptible to enormous improvement by tempering and heat treatment generally. By virtue of this characteristic, steel can be 'automatically' rendered highly resistant to wear and abrasion. Again, vanadium toughens steel; confers upon it great power of resistance to torsional rupture; in a word, endows it with increased 'life' in practical use.

Being a powerful 'medicine,' vanadium is used in small doses only. In engineering steels the maximum proportion required seldom exceeds 0·2%. By its judicious use, combinations are thus made possible which could not be formed without it, and which permit the successful fulfilment of complicated requirements, whether chiefly static, chiefly dynamic, or divided more or less equally between the two classes."—J. Kent Smith, *American Institute of Mining Engineers*.—*London Mining Journal*, Nov. 16, 1907, p. 604. (A. R.)

Manufacture of Tantalum and Niobium.*—"The author, in the *Revue des Produits Chimiques* of November 1, describes the manufacture of metallic tantalum and niobium by the electrolytic method. The electrolyte is composed of a solution of tantalic acid or niobic acid in an organic acid strongly acidulated with a mineral acid in presence of ammoniacal salts. In 3 litres of water 100 gm. of tantalate or niobate of potassium are dissolved. Sulphuric acid is added to this solution until no precipitate is formed. After a rest of some hours, all is filtered and washed. The filtered product is diluted in 1 litre of water and heated to between 70° and 80° C. A hot one-fifth solution of oxalic acid is added until it is dissolved completely. Then the excess of oxalic acid is partially neutralised with ammonia, and, after cooling, sulphuric acid is added, or hydrochloric acid, so that the mass contains about 3%. The bath thus prepared is then electrolysed; the requisite electromotive force is 2 volts, and intensity of current 0·1 to 0·3 amp. The anode is carbon or platinum, and its form rectangular. The cathode may be of the same form, but should preferably be as follows: A very fine set of carbon, platinum, or metallic filaments, not attacked by the electrolyte, are suspended from a horizontal metal rod. This rod is connected with the negative pole, and placed over the bath so that the filaments are almost completely immersed. Then electrolysis is operated. The filaments are covered with metallic tantalum or niobium, and when they have acquired the thickness of 3 to 4 mm. they are removed from the bath and washed in alcohol. These filaments are then wire drawn by the known methods to a thickness of [illegible] to [illegible] mm. and utilised like carbon filaments. The presence of carbon, platinum, or other metals employed for the cathode, or deposit of tantalum or niobium, only constitutes a small waste in manufacture of filaments composed of these metals (niobium and tantalum)."—G. C. BOUHARD.—*London Mining Journal*, Nov. 16, 1907, p. 601. (A. R.)

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* Columbium.

GOLD SLUDGE LOSSES.—"The method of recovery of gold from cyanide solutions by precipitation upon zinc involves, *inter alia*, disturbance of the shavings at intervals between the periods of cleaning-up. The result is that finely divided sludge, in quantity varying with the particular conditions predominating, becomes suspended in the solution in the zinc boxes. Some portion of this matter in suspension does not settle even after the solution has been at rest for some time, and when the box is again started, it has been found that some gold sludge passes out through the exit channel.

Having ascertained that there were no leakages between compartments, wood shavings were placed in the end compartment of each of three zinc boxes and left there for 138 days.

Assays were made frequently of the filtered solutions entering and leaving the compartments of wood shavings, but without in any instance revealing a precipitation of gold.

These shavings were then removed, incinerated, and the product smelted with fluxes yielded bullion containing 11 oz. of fine gold.

From the moment of removal of the shavings from the boxes until the completion of smelting, precautions were taken whereby the possibility of contamination of the product with gold-bearing material was obviated."—E. H. TAYLOR.—*Monthly Journal of the Chamber of Mines of Western Australia*, Oct. 31, 1907, p. 901. (H. A. W.)

MINING.

THE ELECTRIC AIR-DRILL.—"Many features of electrical transmission are undoubtedly convenient and economical; but the direct application of the electric current in rock-drilling has long been a baffling problem; of which, in my judgment, the machine here described has furnished the first, and thus far the only satisfactory solution, by combining the acknowledged advantages of air-driven percussion with the acknowledged advantages of electric power-transmission, while avoiding the acknowledged disadvantages of both systems.

This drill is correctly designated; it is not an electric drill, but more completely an air-drill than any other in existence, because it can be driven by air only and not, like other air-drills, by steam also. Yet, while it is thus distinctly air-operated, the power-transmission is electric, and the sole connection of the drill with the power-house is made by the electric wire, air-compressors and pipe-lines being entirely superseded.

Very near the drill, and connected to it by two short lengths of hose, is a small air-compressor, or, more properly, a pulsator, mounted upon a little truck. This constitutes the entire apparatus of a single drill. Each drill is accompanied by its individual pulsator, and each pulsator is connected to the line of wire from the power-house.

The usual drill-shell is employed, and may be mounted upon tripod, bar or column, according to the work. The drill-cylinder, fitted to slide in the shell, is moved forward or backward by the feed-screw. The cylinder is as simple as can be imagined; a straight bore, having at each end a large opening, and a boss to which the hose is attached. The piston also is plain, much shortened in the body, with a large piston-rod, which has a long bearing in a sleeve-elongation of the cylinder.

Upon a truck is mounted an electric motor, geared to a horizontal shaft, with cranks on each end, which drive two single-acting trunk-pistons making alternate strokes in vertical air-cylinders. One of these air-cylinders is connected by the hose to one end of the drill-cylinder and the other end of the drill-cylinder is connected by the other hose to the other air-cylinder. The air, therefore, in either-cylinder, in its hose and in the end of the drill-cylinder to which it is connected, remains there constantly, playing back and forth through the hose according to the movements of the parts, being never discharged, and only replenished from time to time to make up for leakage. The propriety of calling the apparatus a pulsator instead of a compressor is evident.

The essential details of the cycle of operation will be easily understood. We may assume, to begin with, that the entire system is filled with air at a pressure of 30 or 35 lb. This pressure being alike upon both sides of the drill-piston, it will have no tendency to move in either direction. If, now, the motor, instead of being at rest, is assumed to be in motion, one pulsator-piston will be rising in its cylinder and the other piston will be descending in its cylinder; and, as a consequence, the pressure upon one side of the drill-piston will be increased and the pressure upon the other side will be proportionately reduced, this difference of pressure causing the drill-piston to move and make its stroke. Just before the end of this stroke, the movement of the pulsator-pistons is reversed, and the preponderance of pressure is transferred to the other side of the piston, causing a stroke in the other direction—and so on continuously. The drill thus makes a double stroke, or at least receives a double impulse, for each revolution of the pulsator crank-shaft.*

The drill-cylinder, while generally similar to that of the air or steam-drill, is in many respects quite different; and especially is it remarkable for its simplicity. The usual operating valve-chest; the valve and the complicated means for operating it; the main air-ports and the intricate little passages in and connected with the chest—are all absent, and nothing takes their place. The cylinder-heads are both solid and both fastened securely in place. The split front-head, the yielding fastenings for both heads, the buffers, the springs, the side-rods, etc., of other drills, have all been banished. The cylinder is absolutely plain, with direct openings into the interior, and a boss at each end to which the hose is attached.

The piston also has been simplified. The device for securing rotation is necessarily retained; but the enlargement at the end of the piston-rod, which constituted the chuck and necessitated the split front-head, has been discarded. The piston-rod is much enlarged throughout, and a simple but effective self-tightening chuck is slipped upon the end of it.

The compressor or pulsator-cylinders are likewise simple. There are no valves for either inlet or discharge, and there is neither jacketing nor the slightest need of it. The heating of the air by the compression-stroke is compensated by the cooling which attends the re-expansion of the same air, so that it does not become increasingly hot and heat the parts of the machine with which it comes in contact.

In the foregoing description of the principle of operation I assumed a mean air-pressure of about 30 lb. in the apparatus. It may be asked, how this pressure is secured and maintained. When the pulsator is in operation, the air-pressure in the cylinders alternatively rises above and falls considerably below the mean. At a certain point, indeed, it is below that of the atmosphere; and at this point a little valve is provided, which admits more or less air, until a sufficiency has been provided. At the beginning of operation the influx of air is rapid, so that no time is lost in getting sufficient pressure to begin with. The admission of air and also the apportionment of relative volumes thereof to the two ends of the drill-cylinder are easily adjusted by the operator.

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* See p. 118 of October *Journal*, 1907.

The electric air drill is not troubled by the freezing-up or choking of the exhaust, because there is no exhaust. Moreover, the air does not accumulate moisture, and the temperature does not fall to the freezing-point. Again, air becomes and remains a constant vehicle for the conveyance and distribution of the lubricant. A certain amount of oil being contributed to the system at regular intervals, it would be more difficult to prevent than to insure its reaching every working part.

The length of hose employed seems to be limited to about 8 ft. on each side. The hose may be attached to either side of the drill, but each always to its own end of the cylinder. This length of hose gives all necessary liberty for the location of the pulsator-truck near the drill. The truck (of steel, with flanged wheels) is usually made for the standard 18 in. mine-track, but may be made for any other gauge. Special care in levelling is not necessary, since the pulsator will work at any angle at which the truck can stand.

Either a direct or an alternating current motor may be employed, the latter being preferred because it is a smaller, lighter, mechanically simpler, hardier machine, and more nearly 'fool-proof.' Four different speeds may be obtained with the direct current, and two with the alternating current motor —in the latter case, full speed for steady running and a considerably lower speed for starting a hole or working through bad ground, with immediate transition from the one speed to the other, as required. The controller is on the top of the motor and the operator at the drill can start, speed or stop the motor by simply pulling a cord, this being the only connection. The electrical connection ends at the motor: both the hose and the cord insulate the drill; and the operator is never exposed to the current.

The 5-C electric air drill may be regarded as the full equivalent of the 3·25 in. standard air-drill of any make: of its comparative efficiency something will be said later. The power requirement for this drill is from 18 to 20 ampères at 220 volts, or from 9 to 10 ampères at 440 volts—the electrical equivalent of about 5 h.p. The system being a closed circuit, this is independent of conditions of altitude, which make so much difference with the work of the air-compressor which supplies the ordinary air-drill.

The 4-C electric air drill uses a 3 h.p. motor, and is a much lighter drill throughout, equivalent to a 2·75 in. standard air-drill. Table I. gives particulars of size, weight, etc., of both of these drills:—

Table I. — Dimensions, etc., of Temple-Ingersoll Electric Air Drills:

	5-C.	4-C.
Diameter of drill-cylinder .	5⅝ in.	4·75 in.
Length of stroke . .	8 in.	7 in.
Length of drill — end of crank to end of piston .	45 in.	42 in.
Depth of hole drilled without change of bit . .	24 in.	20 in.
Depth of vertical holes machine will drill easily	16 ft.	8 ft.
Diameter of holes drilled .	from 1·75 to 2·75	1 to 1·5 in.
Strokes per minute . .	425	460
Horse power (at motor) .	5	3

The electric air drill strikes a blow, normally so much harder than that of the air-drill of the same capacity, that it has been found advisable in many cases in 'dressing' the steel bits to make them blunter or thicker, in order to avoid breakage. The practical force of the drill had not been computed before-hand, but was demonstrated in extensive practice and experiment, and the clear and sufficient explanation came later.

The drill-piston, when running at full speed and making a stroke for each rotation of the pulsator crank-shaft, does not strike either head. The hole by which the air enters the cylinder from the hose is located, not at the extreme end, or close to the head of the cylinder, but a certain distance away, so that when the piston approaches the head a portion of inclosed air acts as a cushion, which first checks the piston and then shoots it back. The piston thus starts upon its working-stroke impelled by a certain amount of force which, we may say, has been saved over from the preceding stroke to be utilised for this. The piston after being thus started is driven forward by an air-pressure which increases as it advances, the pulsator-piston being in the attitude of chasing and gaining upon the drill-piston for a considerable portion of the stroke, while in the case of the ordinary drill-piston, driven by a constant flow of air from which it runs away, the pressure must constantly diminish as the piston-speed is accelerated. In the same way by the action of the other pulsator-piston the opposing pressure upon the advancing side of the drill-piston is a diminishing pressure instead of the constant atmospheric resistance, and these combined cause a greater unbalanced difference of pressures upon the opposite sides of the drill, a more rapid acceleration of the piston-movement, and a consequent higher velocity and force at the moment of impact of the steel upon the rock.

Perhaps the most gratifying, and also surprising, revelation of all in connection with the electric air drill is the now indisputable fact that it takes only from one-third to one-fourth of the power, at the power-house, to drive it to do the same work. This is accounted for by the fact that the same air is used over and over, and that all of its elastic force is availed of in both directions instead of exhausting the charge for each stroke at full pressure. There are also no large clearance-spaces to fill anew at each stroke, as these spaces are never emptied.

A valuable feature of the electric air drill, is the ability to yank the bit free if stuck in a hole and immediately continue its work. When the bit of the electric-air drill sticks, the motor and the pulsator-pistons do not stop. If the drill-piston is making, say, 400 strokes a minute, as soon as the bit becomes stuck the piston will receive per minute 400 alternate thrusts and pulls with full force, and nothing could be more effective for freeing the bit than these alternate thrusts and pulls.

When the electric air drill is operated without its own generating-plant, the current being taken from a large power-company, some very low figures are already on record. At Idaho Springs, Colo., a mine-shaft was put down 67 ft. in 24 shifts and the total power-cost was $24 for the entire work.—*Bi-Monthly Bulletin of the American Institute of Mining and Engineering*, Nov., 1907, pp. 991-1000. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

Fuel, Water and Gas Analysis. By J. B. C. Kershaw. 8¾ × 5½. (London: Constable & Co., 1907.) Price, 8s.

"This book deals with the analysis of the raw materials of steam manufacture and of the waste product, and is intended for steam users. The section on waste gases is excellent, and we are glad to see that the author draws special attention to the effects of hydrogen, which, passing off as water, is not dealt with by the ordinary flue gas analyst, yet it carries off a very great quantity of heat in latent form as well as double as much thermometric heat per pound as compared with other waste gas because of its high specific heat."—*Cassiers' Magazine*, Jan., 1908. (A. R.)

Hydraulic and Placer Mining. By Eugene B. Wilson. Second edition. $2·50. (London: Chapman & Hall, Ltd.).

"The author's manual has been largely rewritten and brought up to date. The subject matter now includes the geology and prospecting of claims; the various recovery appliances used in working; the water supply through ditches, flumes, and pipes; the application of water for breaking down and elevating; the costs of placer mining, together with some information on steam shovels, dredges, and dry washers. Dealing exclusively with American placer work, Mr. Wilson devotes two chapters to the Mine Laws of the United States and of the Canadian Yukon. A final chapter supplies tables, formulæ, etc., bearing on hydraulics. The 298 pages are illustrated with diagrams and certain full page half-tones, these latter falling in merits far below the printing and general appearance of a useful little work."—*London Mining Journal*, Jan. 18, 1908, p. 72. (A. R.)

Technical Dictionary in Six Languages. English, French, German, Italian, Spanish, and Russian. By Kurt Deinhardt and Alfred Schlomann. Vol. I., 5s.; Vol. II., 25s. (London: Archibald Constable & Co., Ltd.)

"The absolute indispensability of the technical dictionary was never more clearly apparent than at the present time, when masses of valuable scientific matter are constantly appearing in many publications in all the chief languages of Europe. German and French are known to be necessary languages for the English scientist or technologist, but it is obvious that an ordinary acquaintance with these tongues must leave considerable blanks in the terminology of the arts and industries, and these are supplied by the technical dictionary. Of the work before us, we can only say that it is simply monumental in its size and completeness; nothing of the sort has ever, to our knowledge, issued from the press before. Vol. I., dealing with the machine elements, and metal and wood working, is over 400 pages in extent; while Vol. II., dealing with electrical engineering, extends to just 2,100 pages, which, being of fine, thin paper, are still quite within a conveniently portable compass. The arrangement of the work is admirable. The consultant refers to the index at the end of the volume for the requisite term; he is thence referred to the required page, where he will find the word in all the six European languages. Since the Russian language differs from others in its alphabet, a special consulting index in that language is added. The authors have even found space and enterprise to furnish some thousands of admirably executed wood-cut illustrations of different apparatus. Altogether we have nothing but unstinted praise for the work, which approaches as near to perfection as it is possible to imagine; and, having regard to its value, it is sold at an eminently reasonable price."—*London Mining Journal*, Jan. 18, 1908, p. 72. (A. R.)

Recent Cyanide Practice. By T. A. Rickard. 350 pp. $2. (San Francisco: Mining and Scientific Press.)

This book gives in a convenient form a compilation of the series of articles on cyanidation, as they have appeared in the pages of the *Mining and Scientific Press* between January, 1906, and October, 1907. Description in detail of some of the latest improvements in machinery and devices will be found together with working costs and practice in many parts of the world. Naturally a large portion of this work is taken up by controversial matter, and occasionally a tinge of the acrimony of scientific polemics is revealed to enliven our journey through its thought-stimulating pages. (H. A. W.)

Mining and Engineering and Miner's Guide. By Henry A. Gordon. 615 pp. 10s. 6d. (New Zealand: John Mackay, Wellington; London: Eyre and Spottiswoode.)

This book, now in its third edition and published by the authority of the New Zealand Government, has been undertaken to assist those who have already received a grounding in practical work. It begins with a brief history of gold mining followed by a concise mineralogical account of the metalliferous ores of New Zealand, and passes on to such subjects as the genesis of lode formation, methods of opening out and operating in quartz and alluvial mining, hydraulicing, dredging, explosives and blasting, ventilation, mine surveying, water-flow, piping, dams, water-power, boilers, engines, winding ropes, mill construction and practice, the cyanide process and bridge strains, concluding with a table of contents of cuttings and embankments, a mining glossary, and a good index.

The diagrams and illustrations, though not numerous, are of a practical nature, the text is well supplied with formulæ, and generally the book can be confidently recommended to those who are studying with a view to obtain certificates of competency. (A. R.)

Electro-Analysis. By Edgar F. Smith, Professor of Chemistry, University of Pennsylvania. Fourth edition, 1907. 336 pp. $2·50. (P. Blakiston's Sons & Co.)

"The rapid advance in the technical applications of electro-chemistry has caused the publication of a fourth edition of this book. Not only the older time-honoured methods of electro-analysis are given in detail but also the latest methods, which laboratory research has proved to be accurate. Especial attention has been paid to describing the rapid methods of analysis depending upon the use of a mercury cathode and rotating anode. The length of time consumed in each precipitation is given, which greatly increases the usefulness of the book. It also contains a section upon the determination of cations and anions."—*Mining and Scientific Press*, Dec. 7, 1907, p. 722. (W. A. C.)

METALLURGY OF TIN. By P. J. THIBAULT, F.C.S. Cloth. 12s. 6d. net. (Australia and London: Critchley Parker, in conjunction with Sir Isaac Pitman and Sons, 1901.)

"The distribution of tin ore throughout the world, the principal producing countries being the Malay Peninsula, and the adjacent island of Banca and Billiton, and Cornwall, Australia, and Bolivia is dealt with. The properties of tin—its physical and chemical—and the compounds, form an instructive chapter. The assay and analysis of tin ores and other stanniferous materials are treated comprehensively. For the determination of tin in crude ores, the author recommends the wet processes, but before he comes to this determination, he takes his reader through the dry assay of tin ore, and then goes on to the wet assay, the determination of small quantities of foreign metals in metallic tin, and some alloys of tin, those only which by virtue of the presence of tin, present some difficulties: and the estimation of small quantities of foreign elements in metallic tin. Following on this comes the winning of cassiterite from alluvial and lode matter. This chapter has been kept up to date, only the latest and most suitable tin-saving appliances being described. Included in this are succinct descriptions of hydraulic sluicing, and dredging plants, in operation in various parts of New South Wales. A chief feature is the chapter referring to the preparations of the ore for smelting, for of all the common metals tin is probably the most difficult to refine by purely metallurgical means. The classification of the processes in the preparation of the ore for smelting is—(1) thermo-chemical, followed by mechanical: (2) thermo-chemical alone; (3) electro magnetic: and each of these processes is fully dealt with. In this section the smelting of complex tin sulphide ores which, says the author, is profitable when handled by those conversant with its requirements, discloses some valuable information. The remainder of the book has reference to the refining, then the destannising of stanniferous slags, and the wet metallurgy of tin, which deals more essentially with the handling of waste products from the arts and trades. Altogether the book contains 245 pages, and numerous processes for winning the metal are illustrated."—*The Australian Mining Standard*, Dec. 11, 1907, p. 535. (W. A. C.)

STEAM BOILERS. By W. H. FOWLER. 619 pp. $8\frac{3}{4} \times 5\frac{3}{4}$. 12s. 6d. net. (Scientific Publishing Co.)

"Mr. Fowler has given an exhaustive account of present-day practice in the raising of steam, and of the methods of securing the greatest value from the fuel burnt. Every type of boiler is represented down to the hot-water boiler supplying a heating apparatus; and among the supplementary appliances, stop valves, gauges, feed-water heaters, water-softeners, and superheaters with the many smaller accessories are all treated; while, with regard to the source of the heat, chapters are given upon fuels and the principles of combustion, mechanical stokers, the production of draught and the use of oil in steam boilers.

Each chapter in the book deals with one particular part of the plant, and after explaining the general principles underlying the construction and design, proceeds to discuss typical forms that have been developed in practice. In his description of these, the author has drawn upon his own practical experience and has clearly set out the advantages and drawbacks of the various forms; and one is conscious of an independent judgment being given upon the weak as well as upon the strong points of particular designs.

Speaking generally, the whole discussion of the subject in this book is that of a man conversant with the matter in practice, and while it cannot include each one of countless designs, the book is exhaustive in its presentation of what is typical of the best modern practice. Mr. Fowler has also included a chapter upon the qualifications and duties required from inspecting engineers, as a guide to those who are desirous of qualifying for such positions.

The illustrations, of which there are a large number, may be regarded as one of the many good features of the book."—*Times Engineering Supplement*, Jan. 1, 1908, p. 6. (W. A. C.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 56/08. William Hortor Fletcher. Improvements in mills or apparatus for crushing or pulverising ores, minerals or the like. 17.2.08.

(P.) 57/08. Peter Arthur Walsh. Improvements in linings for tube mills and the like. 18.2.08.

(P.) 58/08. James Garvie (1), Sydney Arthur Cole (2). Improvements in valve motions. 18.2.08.

(P.) 59/08. James Hasking. Improvements in and relating to rock drills. 19.2.08.

(P.) 60/08. James Spry. Improvements in skips or other vehicles for use in combined vertical and incline mine shafts. 19.2.08.

(P.) 61/08. John Slorach. Improvements in valve devices for direct acting engines. 19.2.08.

(C.) 63/08. John Hutchings. Improvements in and relating to rock drills or boring tools. 21.2.08.

(C.) 64/08. Ejnar Alexander Hytten. Improvements in or relating to telegraphonic apparatus. 21.2.08.

(C.) 65/08. Hubert Pearce. Improvements in or relating to self-contained aerating machines. 21.2.08.

(P.) 66/08. Mille & Hill. Improvements in percussive tools. 21.2.08.

(P.) 67/08. James Harrington (1), Robert Farry (2). Improvements in percussive drills for use in machine or hand rock drilling. 22.2.08.

(P.) 68/08. Eustace Moriarty Weston. Improvements in the mounting of rock drilling machines. 24.2.08.

(P.) 71/08. Joseph Stamp. An improved safety catch for mining cages, lifts and such like. 27.2.08.

(P.) 72/08. Eustace F. Wilson. A new or improved preservative for leather belts and the like. 28.2.08.

(C.) 73/08. Thomas Martin Topp. Improvements in stamping and grinding mills. 28.2.08.

(C.) 74/08. Thomas Aitken, M.I.C.E. Improvements in or relating to apparatus for distributing liquids on roads. 28.2.08.

(C.) 75/08. Leopold Benoit de Laitte. Improved carburetting apparatus for internal combustion engines. 28.2.08.

(C.) 76/08. William Lawrence Wildy (1), William Vivian (2). Improvements in pressure operated valves for fluid pressure engines. 28.2.08.

(C.) 77/08. Robert Connolly (1), Frederic Victor William Swanton (2). A new apparatus and improved process for the extraction of salt from brine for the separation and of other matters held in solution in liquids. 29.2.08.

(P.) 78/08. Thomas Holmes. Improvements in means for fixing tappets to the stems of battery stamps. 2.3.08.

(P.) 80/08. John Richard Williams. An improved process for the treatment of refractory ores. 4.3.08.

(P.) 81/08. Reuben Jones. Improvements in percussive rock drills. 5.3.08.

(P.) 82/08. Herbert Helsall Woodford. Improvements in or relating to bottles or receptacles for fermented liquors, aerated liquors and the like. 6.3.08.

(P.) 83/08. James Philip McKeown. Improvements in ore feeding devices for stamp mills. 6.3.08.

(P.) 84/08. Alfred Moggridge Day. Improvements in means for distributing pulp in cyanide tanks and the like. 6.3.08.

(P.) 85/08. John Penhale. Improvements in boring and like machines operating by percussion. 6.3.08.

(C.) 86/08. Walter Harcourt Palmer. Improvements in and relating to explosives. 6.3 08.

(C.) 87/08. Francis Henry Crittall. Improvements in the manufacture of expanded metal and in apparatus therefor. 6.3.08.

(C.) 88/08. Philip Ferdinand Peek. Improvements in centrifugal ore separators. 7.3.08.

(C.) 89/08. Philip Ferdinand Peck. Improvements in centrifugal ore separators. 7.3.08.

(P.) 90/08. Edward Aston. Improvements in candle protecting devices. 9.3.08.

(P.) 91/08 Zilads Holger Lauridsen. Improvements in safety catch mechanism. 10.3.08.

(P.) 93/08. William Mauss. Improvements in self-feeding percussive apparatus. 12.3.08.

(P.) 94/08. Wilhelm Mauss. Improvements in fluid propelled direct acting engines. 12.3.08.

(C.) 95/08. Levi Dee York. Improvements in pulverising mills. 13.3.08.

(P.) 96/08. Phillipus Rudolph Roux (1), George Broom (2), John Percival Robertson (3). Composition for anti-friction grease. 14.3 08.

(P.) 99/08. J. Christ. Improvements in coal washing and ore separating machines and the like. 16.3.08.

(C.) 100/08. Horace George Nichol. An improved slimes separator. 17.3.08.

(P.) 101/08. Herman Franken Schalke. Transportable mining closet. 18.3.08.

(C.) 103/08. Rudolf Spoendlin. Improvements in pumps and apparatus for obtaining motive power. 20.3.08.

(C.) 104/08. Leon Grognot. Process for manufacturing new synthetic india-rubber. 20.3.08.

(P.) 106/08. John Sachs. A new and improved chemical preparation for sweetening purposes and mode of manufacturing same. 23.3.08.

(P.) 108/08. Peter Arthur Walsh. Improvements in the lining of tube mills and the like. 24.3.08.

(P.) 109/08. William John McCann. An improved lever jockey for hauling trucks on mechanical haulage. 24.3.08.

(P.) 110/08. William Alfred Lewis Martin. Furnace for treatment of refractory ores. 26.3.08.

(C.) 112/08. Edward Halford Strange (1), Charles Albert Pim (2). Improvements and apparatus for the manufacture of thin sheets, or foil, or strips, or ribbons, of zinc, lead, or other metal or alloy. 28.3.08.

(C.) 113/08. Charles Hamilton Gray. Improvements in or relating to railway buffers. 28.3.08.

(C.) 114/08. William Harvey Brown. An improved reinforcing bond for buildings. 30.3.08.

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VOL. VIII. APRIL, 1908. No. 10.

Proceedings
AT
Ordinary General Meeting,
April 18, 1908.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, April 18th, Dr. James Moir (Vice-President), in the chair. There were also present :—

33 Members : Messrs. T. Lane Carter, F. F. Alexander, R. G. Bevington, W. R. Dowling, K. L. Graham, A. McA. Johnston, A. Richardson, G. O. Smart, Prof. G. H. Stanley, A. Whitby, H A. White, Prof. J. A. Wilkinson, W. A. Caldecott, S. Beaton, W. Beaver, A. D. Gilmore, F. G. Guthrie, J. I'Ons, H. R. Jolly, J. Kennedy, C. B. Kingston, Hy. Lea, Jas. Lea, W. P. O. Macqueen, G. Melvill, J. T. Milligan, J. F. Pyles, O. D. Ross, C. E. Rusden, Ralph Stokes, W. H. Stout and W. Taylor.

10 Associates and Students : Messrs. J. Chilton, J. Cronin, C. L. Dewar, D. B. Donovan, A. L. Edwards, J. H. Harris, Jas. Hawthorn, R. W. Leng, W. E. Thorpe and L. J. Wilmoth.

6 Visitors and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting as published in the *Journal*, were confirmed.

NEW MEMBERS.

Messrs. W. Taylor and G. Melvill were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that the candidates for membership had been duly elected, as follows :—

CURNOW, ROBERT, Nigel G. M. Co., Ltd., P. O. Nigel. Mine Manager.
ORR, JOHN, B.Sc., M.I.Mech.E., A.M.I.C.E., Transvaal University College, P. O. Box 1176, Johannesburg. Professor of Mech. Engineering.
SIMPSON, FREEBAIRN LIDDON, Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Amalgamator.
THOMAS, JOHN, West Bonanza G. M. Co., Ltd., P. O. Box 137, Klerksdorp. Cyanide Manager. *(Transfer from Associate Roll.)*

The Secretary announced that the following gentlemen had been admitted as Associates by the Council since the last general meeting :

BROWN, WILLIAM SINCLAIR, Lebong Soelit, Ketaun, Sumatra, Dutch East Indies (via Benkoelen). Metallurgist.
KAEDING, HENRY B., 1421, Winfred Street, Los Angeles, Colo., U.S.A. Mining Engineer.
LOFTS, HORACE F., St. Stephen's Lodge, Trowbridge, Wiltshire, England. Metallurgist.
NELLMAPIUS, ALBERT PHILLIP, P. O. Box 6499, Johannesburg. Mining Engineer.
SHORT, ARTHUR REGINALD, New West Bonanza G. M. Co., Ltd., P. O. Box 137, Klerksdorp. Assayer.
WISEMAN, EDWARD JAMES, Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Foreman Trammer.

And as Students :

GAISFORD, ARTHUR, Transvaal University College, P. O. Box 1176, Johannesburg.
ELOFF, STEPHANUS JOHANNES PAULUS, Transvaal University College, P. O. Box 1176, Johannesburg.
MOON, CYRIL LESLIE, P. O. Box 6695, Johannesburg.

All are Students of the Transvaal University College.

GENERAL BUSINESS.

The Secretary : I wish to bring to your notice the visit to the Nigel, which had to be postponed, until the 9th of May, providing we get a sufficient attendance. I would ask members, if they can possibly take part in this excursion, to send in their applications as soon as possible. We have to guarantee £32 10s. for the special train. It is useless going down by the ordinary train and coming back by the Natal Mail because that will only give us about an hour at the mine. A limited number of ladies may accompany the party, and the Managing Director and the General Manager of the mine have gone to considerable trouble to make the visit a success.

Prof. J. A. Wilkinson then proposed a very hearty vote of thanks to the gentlemen who formed the sub-committee for the Annual Dinner, which was carried unanimously.

A FEW NOTES ON STAMP MILLING.

By W. H. Jane (Associate).

In placing these notes before this Society, I am actuated with a desire not to impart, but to obtain, information during the discussion which I hope will follow.

It is, I believe, a generally accepted theory that five stamps are run in one battery, to complete as nearly as possible a cycle, thereby obtaining a steady load on the driving belt; but apparently the fact is often lost sight of that it is also intended to produce the action of a cycle in reducing ore. In other words, the five stamps ought in falling on the dies to act as nearly as possible in the same manner as a roller which exercises the pressure of a battery of stamps, would in passing over a given quantity of ore. If it were possible to produce a metal of sufficient resistance which would wear evenly along the whole of its face, it would no longer be necessary to reduce our ores by means of stamp mills, with their serious loss of efficiency—by applying the power consumed to the point at which it does its work, *i.e.*, between the shoe and the die.

If we accept this theory, we must set our cams on the cam shaft so as to get an even distribution of ore along the crushing faces, and not (as is often seen) Nos. 1 and 5 falling on a much greater quantity of ore than Nos. 2, 3, and 4. In this case it is necessary to run Nos. 1 and 5 with a longer drop than Nos. 2, 3, and 4. This practice at once destroys the cycle and continuity of action, impedes the discharge of crushed ore from the mortar, puts an uneven strain on the driving belt, and causes a loss of power at Nos. 2, 3, and 4. The fact that the centre stamp is called upon to feed ore into the mortar, and Nos. 2 and 4 to distribute it to Nos. 1 and 5 upsets the theory but only to a small extent; and this can be remedied, either by having slightly heavier centre shoes, or a slightly longer drop, but all stamps should be run at the longest possible drop, and should not be shortened more than the maximum amount of wear of the shoes and dies per day, at any one time. The practice of shortening stamps by inches should on no account be allowed.

Another practice I do not agree with, is running the stamps at over 100 drops per minute. On a hard ore, stamps running at 98 drops per min., with a maximum drop, will do more work with less wear on the mill than stamps running at 100 drops per min. This is caused by the increased force of the blow, owing to increased length of drop and the laws governing falling bodies. In short, the exact speed required is that which allows the stamp to fall the full length of the cam minus the necessary clearance, the stamp to be lifted immediately it has expended its force. In crushing soft ores it would be necessary to strike oftener and with less weight, or increase the diameters of shoes and dies, and crush a greater quantity at each blow, though in the latter case we have a distinct loss, owing to waste space in the mortars, which would have a tendency to allow the ore to pack between the shoes and the liners, which again would interfere with fine feeding. For this reason, I think, it is not desirable to increase the diameters of shoes and dies, even when installing heavier stamps, as I do not think we are as yet very near the point at which increased diameters will be essential. To feed perfectly it is necessary to have gear to operate the Challenge feeders with the least possible movement of the rocker bar. That is, the collar must not strike the rocker bar until the stamp has almost finished its stroke. In the old suspension gear, even when in fair order, the bar had to move a fair distance before operating the feeder. The use of springs to pull back the pawl or friction discs is objectionable. Taken singly, the pull of each spring amounts to little, but in a large mill we have a considerable amount of power expended uselessly. The rocker bar designed by Mr. G. O. Smart, and installed in several of the Consol. Goldfield's mills, makes an almost ideal rocker bar, its perfect balance using up very little power. It feeds from the stamp in proximity to the tappets, and conveys motion by means of a hemp rope to the feeder gear. If the rope is wound spirally with hoop iron, 1 ft. or so above and below where it passes through the cam platform it lasts a very long time and gives no trouble. With tube mills running in conjunction with gravity stamps, it is highly undesirable to make the stamps do too much work, since at each blow struck the stamps liberate a lot of stuff which is not expeditiously expelled from the mortars, and though it is admitted that a tube mill is not an economical primary grinder, yet taken in conjunction with gravity stamps, it will be found we do not reach a very high standard of efficiency when using too fine screening. There are enough tube mills installed on the Rand at present to enable us to discover by actual practice the percentage of work which stamps and tube mills should do, if members will only give us the benefit of their experience.

The Chairman: This is a paper by one of our younger Associates, and I am glad to see that he is not afraid to face the Society in the excellent way he has. I hope that some of our older members will discuss the paper and possibly supply Mr. Jane with some of the data which he asks for.

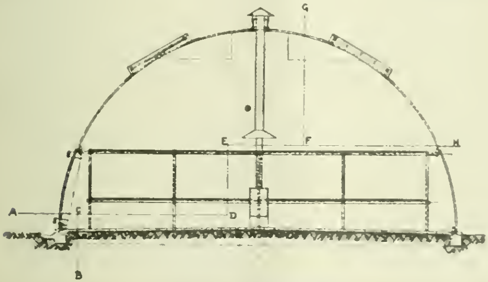
NEW TYPE OF NATIVE COMPOUND BUILDING OF ALL METALLIC CONSTRUCTION.

By C. B. Kingston, B.A., B.A.Sc. (Member).

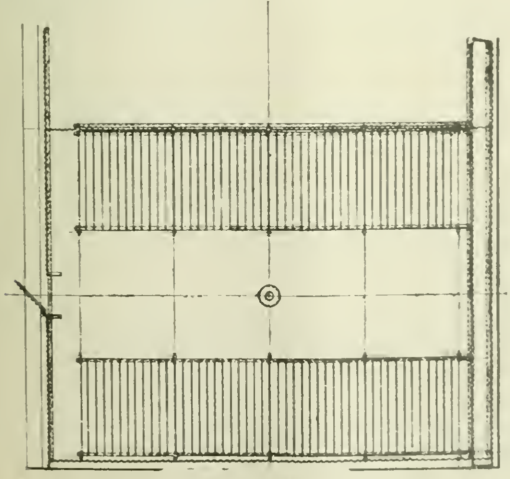
Having occasion recently to erect a new compound for the Great Eastern Colliery, at Springs, the writer set about designing a cheaper type of building than the one generally used for housing natives.

It was first decided to erect a wood and iron building. To save framing, it appeared that a curved iron roof was desirable. To lessen the framing in the walls, it was sought to reduce these to a minimum, and as a natural consequence, it was finally decided to eliminate the walls altogether. This was done, and the building, as erected, consists simply of a corrugated-iron roof resting on a magnesite concrete floor.

The section of the building is a semi-circle with a radius of 14 ft. The radius was fixed by the Regulation that provides that in calculating the cubical air space of rooms with two tiers of bunks, no allowance is made for space above 14 ft. from the floor. The rooms are therefore 28 ft. long and 14 ft. high. They are made 21 ft. wide to accommodate 32 boys with 200 cub. ft. of air space per boy.



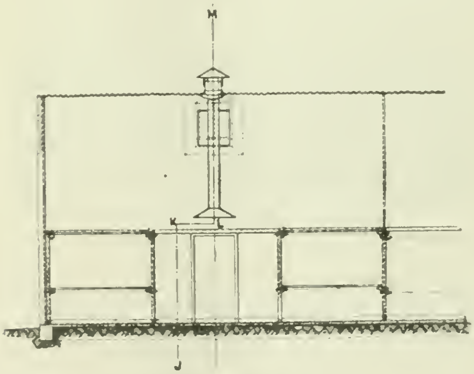
END ELEVATION.



PLAN ELEVATION.

The building is constructed of good 24-gauge iron, curved in the rolls used in the manufacture of ordinary water tanks. The sheets are single riveted on the curved joints, and double riveted on the horizontal joints. The edges are attached by clamps to a $1\frac{1}{4}$ in. black iron pipe bolted down to the concrete floor. The floor has a slope of 3 in. in the 14 ft. from the centre of the rooms to the sides. The bunk frames are of $1\frac{1}{4}$ in. pipe stayed with $\frac{5}{8}$ in. iron bolts. A stringer of similar pipe runs the length of the building on either side above the doors, and is bolted to the top members of the bunk frames, which are thus tied together. The corrugated iron of the roof is also clamped to these stringers and supported by the bunk frames. This reduces the span of the unsupported curved iron roof to 24 ft. The rooms are separated by partitions of 24 gauge corrugated iron attached to the roof by angle pieces of plain galvanised iron of the same gauge. Spaces are left to allow the air to circulate freely through all the rooms. To give extra stiffness, the two central sheets of iron in each partition are extended, and securely fastened to the roof.

The door and window frames are easily attached to the corrugated iron in a perfectly water tight manner. The building is fitted with regulation ventilators and stove pipes. Small spaces between the floor and the roof are left at regular intervals for intake ventilators. These also act as scuppers to run the water off into the outside drains when the floors are hosed down. If the pipes to which the roof is attached are fastened to the floor with clamps, instead of through bolts, they can be fitted with stop cocks in each room and can be used to supply a water service for washing and fire purposes. As there is no framework in the building except the pipe on the floor and the pipe tying all the bunk frames together, there is a minimum lodgment for dirt, and therefore the building is very easily kept clean.



SIDE ELEVATION.

Since its occupation, it has been tried in hot weather and in cold weather, and has been exposed

BUNK FRAMES ERECTED ON MAGNESITE CEMENT FLOOR AND USED AS SCAFFOLDING FOR ERECTING THE ROOF.

to heavy storms of rain and wind, and under all circumstances has proved entirely satisfactory. Eighteen bed-rooms complete, to house 576 natives, were erected at a cost of £2 12s. per boy. The kitchen and hospital were transferred from the company's old compound, and therefore the cost of these additions, if of the same construction as the rest of the building, cannot be given exactly. A compound of this construction, however, can be erected complete with hospital, kitchen and bathroom for from £2 17s. 6d. to £3 10s. per native, according to the number of boys to be housed. The ordinary well equipped compound of the existing type costs, complete, from £7 10s. to £10 per native. The new type of building of all metallic construction costs therefore less than one-half of the present compounds. The roof of the ordinary compound contains 70% of the quantity of iron in the whole of the new type of building, and as the ordinary roof requires wooden trusses, ridging and guttering, it costs almost, if not quite, as much as the entire iron sheathing of the new building. In point of durability, if used on a deep level mine with a long life, the new type of building compares favourably with the old compounds. The interior iron work of the new type is permanent, while the curved roof will probably last a little longer than the usual pitched roof, and can be renewed for about the same cost.

VIEW OF COMPOUND FROM INSIDE.

If erected on a mine with a prospective life of 25 years, the corrugated iron sheathing might have to be renewed once, which would be the case with the roof of the ordinary compound also.

In the present brick compounds an appreciable amount of money is spent in 12 years on the maintenance and cleansing of the walls, and this expenditure would tend to increase rather than diminish during the second half of its life. In the new type of building the cost of cleansing the walls is much less than in the old, and when the iron sheathing is renewed, the building is fresh and new throughout, which is a distinct advantage from an hygienic point of view.

The difference in first cost between the new type of building and the old is so great, that the interest t for 12 years on the money saved will more than cover the cost of renewing the corrugated iron.

Dr. Sansom, lately District Medical Officer of Health for the Witwatersrand, when examining the plans of the new building, was good enough to express strong approval of it from his point of view, viz., the health of the natives, and on his recommendation the Native Affairs Department propose to adopt this type of building for the native rest houses to be erected on the Natal border. In its general appearance the new compound is neat and workmanlike. It has at least as strong a claim to architectural beauty as the old compound, and the boys appear to like it, perhaps because the curve of the roof suggests the outlines of their native grass huts. The merits claimed for this new type of building are its low cost and its exceeding cleanliness.

The accompanying drawings and photographs show the details of construction.

The Chairman: This is a very interesting paper. I think the new invention seems to be quite excellent—apart from the question of saving money. It seems to be likely to be beneficial to the health of the boys: and I may say also I thought the building had some claim to beauty when I saw it last week. I move a hearty vote of thanks to the author.

Mr. T. Lane Carter: I beg to second the motion. There is only one point that strikes me. Mr. Kingston spoke of the new deep levels, and the suitability of these compounds for such mines is only a question of durability. Suppose you have a deep level mine lasting 25 or 30 years, as they will do, how would a compound of this type come out after such a period?

Mr. C. B. Kingston: I think you would find that in the course of 25 or 30 years you would have to renew the original iron roof on the ordinary compound, and if you work it out there is not very much more iron in this semi-circular section than there is in the pitched roof of an ordinary compound, and the new type is so much cheaper that it would pay you to renew the iron sheathing at least once and even twice in that period

Mr. F. F. Alexander: I should like to know how the doors of the compound are worked. I take it the door would open outside, and if so, how does this door prevent leakage from the circular roof running into the hut itself?

Mr. C. B. Kingston: The wooden frame of the door is inside, on the curve running from the floor up to the stringing pipe that runs across the top of the bunks. The door lies flat in the frame, and there is a small ridge projecting above the top which keeps out the water completely. We have had several heavy rainstorms since this compound was erected, and no water has found its way in at all.

LABORATORY TESTS ON THE USE OF COARSE AND FINE LIME FOR CYANIDING.

By W. J. Sharwood, Ph.D., R.S.M. (Associate).

The object of these experiments was to ascertain the relative rapidity with which commercial lime, in varying states of division, would be dissolved when distributed through a charge of inert sand subjected to the action of percolating water or cyanide solution—the proportions of water, sand, and liquid being practically the same as prevail in the leaching of tailings at the cyanide plants of the Homestake mine. This involves the addition of about 3·5 lb. of lime per ton of sand, through which solution percolates at the rate of about 1 ton of solution in 5 days for each ton of sand.

As the tailings actually treated are pyritic, and would develop acidity sufficient to decompose a considerable proportion of the free lime, a clean quartzose sand (over 98% SiO_2) was used in the tests. This was crushed through a sieve of 40 meshes per linear inch (aperture ·015 in.) and was washed free from slime. One gram of lime was mixed with 575 gm. of sand, placed in a narrow percolator, and leached with distilled water (or in a few cases with stock solution from the cyanide plant) at as nearly as possible a uniform rate of 115 c.c. per day. The effluents were removed and measured at intervals, and the alkalinity of each was determined by titration with sulphuric acid.

A sample of a few pounds weight of the lime lumps, as delivered at the cyanide plant, was crushed and sized by passing through sieves ranging from 10 to 200 meshes per linear inch, the average apertures of which approximated ·074, ·033, ·015, ·011, ·006, and ·003 in. respectively. All coarser than 10 mesh was rejected The quantities obtained from the various grades differed greatly, very small amounts being found between 40 and 200 mesh, and these particles being comparatively hard and gritty. Another sample was taken from a lot of lime which had been stored for some months and had become air-slaked. All but a minute fraction of this passed a 200 mesh screen, with the aid of a little gentle rubbing. Each sample was analysed by the usual proximate method. The "available" lime was also determined in two ways. (1) Two grams were treated with 1,000 c.c. water containing 20 gm. sugar for 2 hours, with occaional very gentle shaking, to avoid breaking up the particles, with the object of obtaining, as far as possible, surface action only. (2) A like amount, weighed out after grinding to impalpable powder, was mechanically shaken with the same proportions of sugar and water for 3½ hours, to get the maximum solvent effect. Bottles of 2,500 c.c. capacity

Table I.—Analyses of Sifted Lime.

Lot	1	2	3	4	5	6	7
Mesh of sieves	10 – 20	20 – 40	40 – 60	60 – 100	100 – 200	200 +	Air-slaked
Dimensions of Apertures Ins.							
"Through"	·074	·033	·015	·011	·006	·003	·003
"On"	·033	·015	·011	·006	·003	0	0
Percentage Composition—							
Loss on ignition ...	3·58	4·13	4·14	4·23	4·06	3·95	22·2
Insoluble	2·61	2·52	2·51	2·98	3·85	2·92	1·83
Fe_2O_3,Al_2O_3	1·42	1·18	0·85	1·30	1·40	1·88	1·12
MgO	2·32	2·29	2·23	2·31	2·13	1·98	1·36
CaO (total)	89·02	88·53	88·81	88·04	87·34	88·18	72·90
Total determined ...	98·95	98·65	98·54	98·86	98·78	98·91	99·41
Soluble CaO by Sugar—							
(I.) 2 hours, not ground...	60·5	58·8	58·3	66·1	76·7	82·9	64·5
(II.) 3·5 hours, ground fine	81·9	80·3	81·4	80·7	77·5	82·0	62·0

were used in all cases (see Note II.). The analytical results are given on previous page in Table I.

The method of recording the results is illustrated by a typical log, shown in Table II. In some tests the effluent samples were taken at more frequent intervals, but this was found unnecessary except in the case of the coarser fractions, or near the beginning and end of a run.

TABLE II.—LOG OF LIME PERCOLATION TEST.

Grade of lime, 40 – 60 mesh, ·015 – ·011 in. Leached with water.
Weight, 1·0 gm. = 0·888 gm. total CaO or 0·814 gm. available CaO.

Volume Effluent c.c.	Cumulative Volume of Effluents c.c.	Volume $N/5$ Acid for Effluents c.c.	Cumulative Volume $N/5$ Acid c.c.	Cumulative CaO in Effluents Grams.	Percentage of CaO in Charge Dissolved. Total.	Percentage of CaO in Charge Dissolved. Available.	Alkalinity of Effluent (Note III.).
35	35	5·5	5·5	·031	3·5	3·8	31·4
110	145	25·4	30·9	·173	19·6	21·3	46·2
110	255	24·5	55·4	·311	35·	38·2	44·5
110	365	24·3	79·7	·446	50·2	54·8	44·25
110	475	21·4	101·1	·566	63·7	69·6	39·0
100	575	16·4	117·5	·658	74·1	80·8	32·8
125	700	13·1	130·6	·731	82·3	89·9	21·0
100	800	3·1	133·7	·749	84·3	92·0	6·2

As it was obvious that, weight for weight, the slaked lime carried only about three-quarters as much available CaO as the fresh, a special run was made using one-third more, or 1·33 gm. of slaked lime to 575 of sand, to make the amount of "available" practically equal to that used in other tests.

Fig. I. shows the variations in alkalinity of the effluents at various stages of the leaching. The abscissæ represent times or volumes of effluent. The ordinates representing the average alkalinity of the various effluents are erected at the middle points of the periods (or volumes) to which they correspond. For the sake of clearness, when represented on a small scale, some of the curves corresponding to intermediate grades of lime have been omitted in this series, and also in Fig. II.

Fig. II. shows in a similar manner the cumulative amounts of lime dissolved and removed during the progress of percolation, expressed as percentages of the total available CaO present in the charge at the commencement of the tests. The ordinates corresponding to percentage (taken from column 8 of log) are erected at the points representing the ends of the periods to which they refer.

The complete series of curves, indicated in Fig. II., were plotted on a large scale, and the percentages removed at the end of each day were scaled off, and are tabulated in Table III.

With regard to the results omitted from the figures, the curves corresponding to 40—60 and 100—200 mesh, and also that for "slaked lime 1·33 gm.," all approximated closely to the curve representing 60—100 mesh which appears in Figs. I. and II.

It will be noted that the curves in Fig. I., representing the alkalinity of cyanide solutions percolated through a mixture of lime and sand, are nearly parallel to those obtained with water and lime of the same grades, the solvent power of the cyanide solution for CaO being substantially the same as that of water.

TABLE III.

Number of days leaching … Tons effluent per ton of sand	1 0·2	2 0·4	3 0·6	4 0·8	5 1·0	6 1·2
GRADE OF LIME USED. Mesh of sieves used, per linear inch.	Percentage of available lime leached out.					
10 – 20	6·7	19·4	33·0	44·6	54·5	—
20 – 40	13·4	31·3	48·8	63·6	77·5	88·0
40 – 60	16·7	34·0	51·7	67·5	80·6	89·0
60 – 100	16·9	35·0	53·9	69·8	83·5	90·1
100 – 200	19·4	37·8	57·6	74·0	88·5	95·5
Through 200	21·0	40·0	60·0	74·0	85·6	—
Slaked lime, lot 1 } Through 200	25·0	47·3	70·2	84·0	92·0	—
" " lot 2	22·1	44·0	67·0	83·6	92·0	94·5
" " lot 3 } Weight increased 33% to give equal CaO	18·0	36·0	55·2	70·6	80·6	89·6

In a general way, the result of this series of experiments confirms the experience obtained in the Homestake practice, viz., that the most desirable condition is to have the lime largely in particles which are neither very coarse nor extremely fine. If much of it is in granules, *e g.*, coarser than a 20 mesh (linear) screen (·033 in.) these grains will in great part remain undissolved at the end of the treatment. Previously slaked lime, and particles finer than 100 mesh (·006 in.) yield about 90% of their available alkali in 5 days' leaching, but yield it at a rate which is not sufficiently uniform. Such highly comminuted lime gives too high an alkalinity to the cyanide solution at the earlier stages of leaching, a condition which (with the pyritic ore in question) is detrimental to the solution of gold, it having been found that a "protective alkali" higher than 0·3 lb. per ton of solution gives decidedly low extractions. Another point in favour of a granular lime is the fact that the acidity of these tailings is distinctly cumulative, there being 5 to 10% of sulphide of iron present, a large proportion of which is pyrrhotite. The experimental results were obtained with an inert charge; under working conditions, with an acid charge, and especially with one constantly developing acidity, there is no doubt that the available CaO added would have disappeared more rapidly—in spite of the fact that the whole of it was actually introduced into the charge, which is not the case in actual work. With too fine a lime a very large proportion either dissolves in the water conveying the tailings, or remains suspended in it while the vat is filling, and so passes off at the peripheral overflow. In some cases this overflow water has been found to be a nearly saturated solution of caustic lime. While this would suffice to neutralise any original acidity existing in the tailings, it is, of course, useless as regards that which may develop later.

The lime used at the Homestake plants is burned in continuous wood-fired kilns at the company's quarry, and loaded into railroad cars. At No. I. plant the cars are unloaded into bins, from which the lime is conveyed by a gravity wire rope system to bins in the plant. Here it is crushed through a jaw crusher to about 1½ in. size, and sufficient is weighed out for 2 hours' supply This is dumped into the hopper of a Challenge feeder supplying the mortar of a one-stamp mill, having one screen, 11 in. × 14 in., and fed with a small stream of water, which carries the lime to a launder where it mixes with the previously classified tailings on their way to the distributor and leaching tank. A further addition of lime is made during the progress of leaching, by spreading 200 lb. over the surface of each charge of 600 tons. If, however, the alkalinity of the earlier effluents is unduly high in comparison with the percentage of cyanide (or in other words, if the protective alkali is excessive) this top-lime is omitted.

Note I. Analyses of Lime.—The values quoted for "insoluble" include the total silica. The mixed oxides, as weighed, contain approximately equal proportions of alumina and oxide of iron, together with a very little oxide of manganese.

Note II. Estimation of Available Lime.—The method followed is based on that of Mr. G. W. Williams,* using a 2% sugar solution. I filter a portion of the liquid through dry paper as soon as possible after it has settled comparatively clear, and at once measure out the volume to be titrated into a beaker or flask. If titrated at once phenolphthalein is generally used as indicator. Identical results are, however, obtained with methyl orange, if the liquid has been filtered, and this indicator is distinctly preferable, as it may be used after a considerable amount of carbonate has been formed by exposure to the atmosphere, a slight excess of standard sulphuric acid being added to decompose any carbonate thus formed, and the excess of acid being titrated with standard soda.

Since writing these notes I have received the November and December numbers of the *Journal*, containing criticisms of the sugar process. My own experience has been that a 2% solution of sugar, 1,000 c.c. to 2 gm. of sample, gives very satisfactory results; its solvent power is ample as regards finely divided CaO, and its effect on other calcium compounds appears to be little if any greater than that of pure water. Of course the object of this determination (from the cyanider's point of view) is to establish the amount of alkali available to neutralise weak acids, and in this case there is no doubt that some of the calcium silicates have a slight solubility and can exercise a feeble protective effect. Calcium carbonate in the original sample does not appear to influence the results of this determination in the slightest degree.

Note III.—The "alkalinity number" used at these plants corresponds to the number of cubic centimetres of decinormal acid required to neutralise 100 c.c. of solution. This system has been in use for over 10 years and is very convenient, as it does not presuppose the existence of any particular compound, such as CaO, in expressing the alkali present, while it can at once be translated into terms of percentage of CaO or NaOH by multiplying by the equivalent weight of either compound.

1 unit = ·0028% CaO = ·056 lb. CaO per ton.
,, ·0056% KOH = ·112 lb. KOH ,,
,, ·0040% NaOH = ·080 lb. NaOH ,,
,, ·0065% KCN = ·13 lb. KCN ,,

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* See this *Journal* Vol. v., April, 1905, p. 253; also vol. viii., pp. 37, 124 and 206.

A saturated solution of lime in water has an alkalinity of about 45.

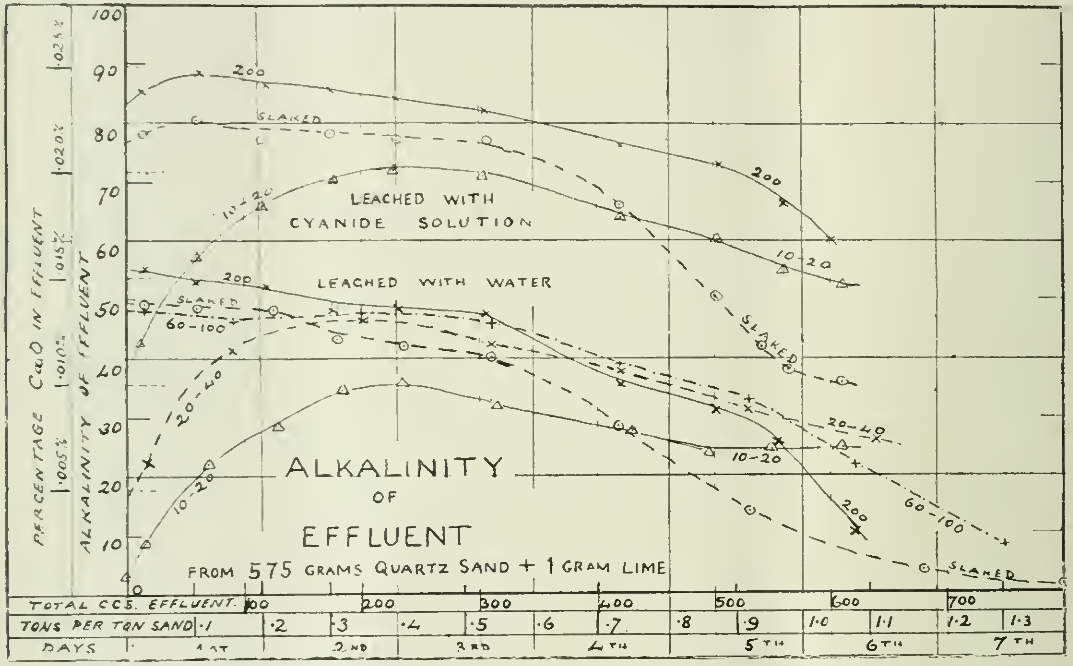
In a cyanide solution of known percentage KCN and alkalinity estimated by above standard, the "protective alkali" in pounds CaO per ton, can be calculated by the formula:—

$$\cdot056a - 8\cdot6k = \text{pounds CaO per ton},$$

where a = alkalinity by above standard, and k = percentage KCN.

In daily work the solution man takes samples of 20 c.c., adds 2 or 3 drops of 1% phenolphthalein solution, and titrates with $N/5$ sulphuric

FIG. I.



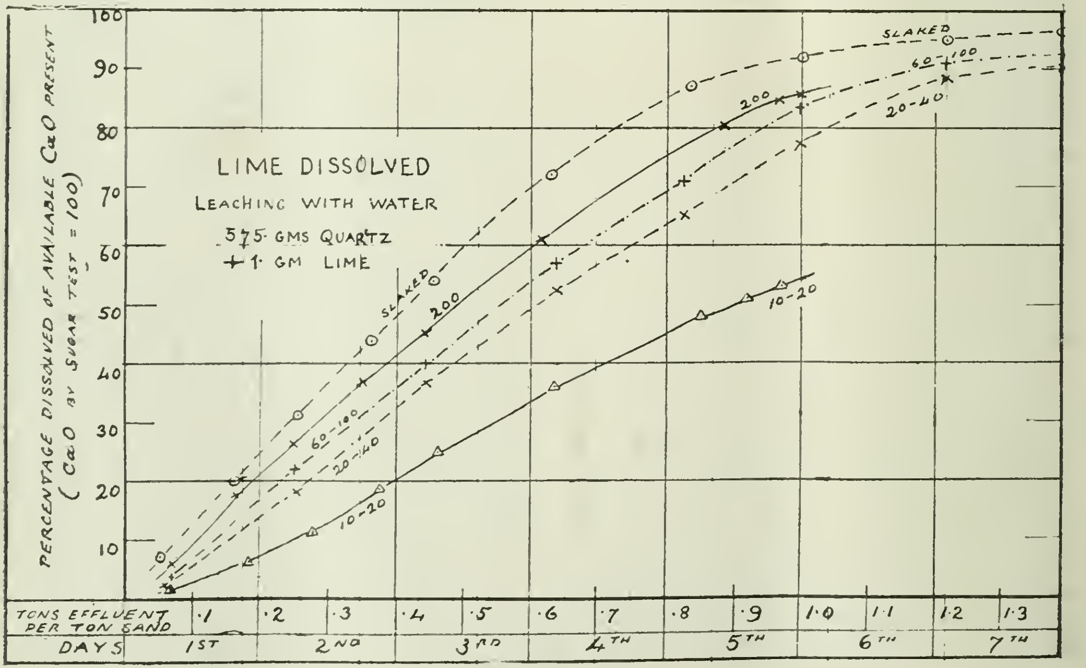


FIG. II.

acid. Each tenth of a cubic centimetre of acid used represents one unit of alkalinity. When the "protective alkali" value is required in pounds lime per tons it is read from a table calculated from the formula above given. Synthetic tests have shown that this method, while not by any means strictly accurate in the presence of double zinc cyanide, is liable to no greater errors than the methods based on titration after precipitation with silver nitrate.

Note IV. Preparation of Standard Sulphuric Acid.—Strong acid is diluted to give a stock solution of medium strength, slightly greater than 4N. Exactly 100 c.c. of this is diluted to 2,000 c.c., and the H_2SO_4 in a sample of this diluted acid is estimated gravimetrically with the greatest care. If, for instance, it proves to be 0·2055 N, we must then dilute 100 c.c. of the stock acid up to 2,000 and add 55 c.c. more, in order to get standard acid of exactly 1/5 normal strength.

Mr. W. A. Caldecott: I have much pleasure in proposing a very hearty vote of thanks to Mr. W. J. Sharwood for his able and instructive paper upon a subject that concerns so many of us. The author has done much valuable research work upon the chemistry of cyaniding, and has been long associated with Mr. C. W. Merrill, whose name and historical work at the Homestake will be familiar. In explanation of a portion of the paper, I may say that Homestake practice consists in separating sands from slimes in the tailings pulp by means of a large number of conical classifiers. The district being blessed with an abundance of water, and ample fall having been secured by locating the sand plant a long way further down the creek than the mill, it is possible to deliver a sand pulp to the vats, in which the sand is both collected and treated, almost free from slimes. It is to this sand pulp underflow from the cones that lime is added before entering the distributors, so that it is deposited uniformly throughout the sand charge. The sand contains a good deal of the reducer pyrrhotite, and lime is required to neutralise the oxidation products of this mineral during treatment. Great importance is attached to aeration of the charge during treatment in the vat in which it is collected, and hence every attention is paid to preventing slimes being deposited with the sands, as free access of air into the interstitial spaces between the sand grains is essential to secure "envelope aeration," as it is termed by Mr. Merrill.

Among other interesting features at the Homestake are the very large number of light 900 lb. stamps employed, crushing fine with a 10 in. discharge and a high water ratio. A very high percentage recovery by amalgamation is secured on an extremely large plate and which is dressed once daily.* Regrinding of the coarser sand is now being tentatively introduced, both grinding pans and a tube-mill being employed.† The satisfactory working of Mr. Merrill's large filter-press plant, in which some 1,600 tons daily of very poor slimes are collected, treated, washed and discharged with water without opening the press, has already been described in our *Journal*, as likewise the method employed for efficient zinc dust precipitation.‡

It is to be hoped that Mr. Sharwood's paper is but the first of series detailing the remarkably successful results, achieved by the enterprise and skill of Mr. Merrill and his band of able lieutenants, in solving the problems presented in the treatment of an enormous tonnage of low grade ore handled at one of the greatest gold mines in the world.

The Chairman: I am sure we all wish to thank Mr. Caldecott for his valuable explanation, and accord at the same time a hearty vote of thanks to Dr. Sharwood for his paper.

THE LABORATORY: ITS ECONOMIC VALUE.

(*Read at October Meeting, 1907.*)

By A. McArthur Johnston, M.A., F.C.S. (Member).

REPLY TO DISCUSSION.

Mr. A. McA. Johnston: The criticism evolved by my paper has brought forth complimentary remarks on the one hand and useful opinions and controversial matter on the other. To the latter we can confine our attention, and I am pleased to note the strong attitude taken up by all the speakers in advocating the recognition of the testing laboratory as a profit saving, as well as a scientific institution. It would seem, however, that some of the remarks should have been delivered before our confrères, the mechanical engineers, where, I am sure, they would have brought forth anything but laudatory rejoinders. Still, it is pleasing to note that the testing department is so much thought of by many engineers, as is evidenced by the substantial token afforded to Mr. H. C. Behr for the reply to the criticism on his original paper. Actual buying of coal on the stipulations outlined by him should soon be an accomplished fact, and its concomitant saving in steam produced will then be recognised.

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* See this *Journal*, p. 243, March 1905.

† See *The Mining World*, p. 541, March 28, 1908.

‡ See this *Journal*, p. 24, July, 1907; also *Mines and Minerals*, p. 432, April, 1908.

The method of testing the volatile matter in coal samples does not altogether meet with the approval of Prof. Stanley or Mr. Croghan. In this I can bear with them, for it is not absolutely correct, and I would be only too pleased if money could be obtained to purchase a platinum muffle, crucibles and trays as well as erect a gas plant, so that we could conduct determinations of coal and ash samples averaging, at certain times of the month, from 10 to 20 a day. What I was outlining rather was a practical method which is used by us and could be adopted by any well equipped assayer on these fields. With uniform apparatus and careful manipulation it will be found that the results obtained are accurate enough for all practical purposes, and could well be used for the basis of buying coal supplies. In the calculation of the evaporative power by means of a constant factor, I am afraid my experience is not consistent with that mentioned by Mr. Croghan. To illustrate my point, I will give the proximate analyses of four samples submitted to us and chosen indiscriminately from four different coal producing districts :—

	(1)	(2)	(3)	(4)
	%	%	%	%
Moisture ...	2·20	1·80	1·00	5·48
Volatile matter	18·42	27·34	19·12	19·83
Fixed carbon ...	62·33	60·38	69·76	62·10
Ash	17·05	10·48	10·12	12·59
Evaporat. factor Mahler bomb	11·94	13·48	13·88	11·76
As calculated by Mr. Croghan	11·30	12·28	12·44	11·47
Difference ...	– 0·64	– 1·20	– 1·44	– 0·29

These results sufficiently show the difficulty, if not the impossibility, of calculating heat values from the proximate analysis.

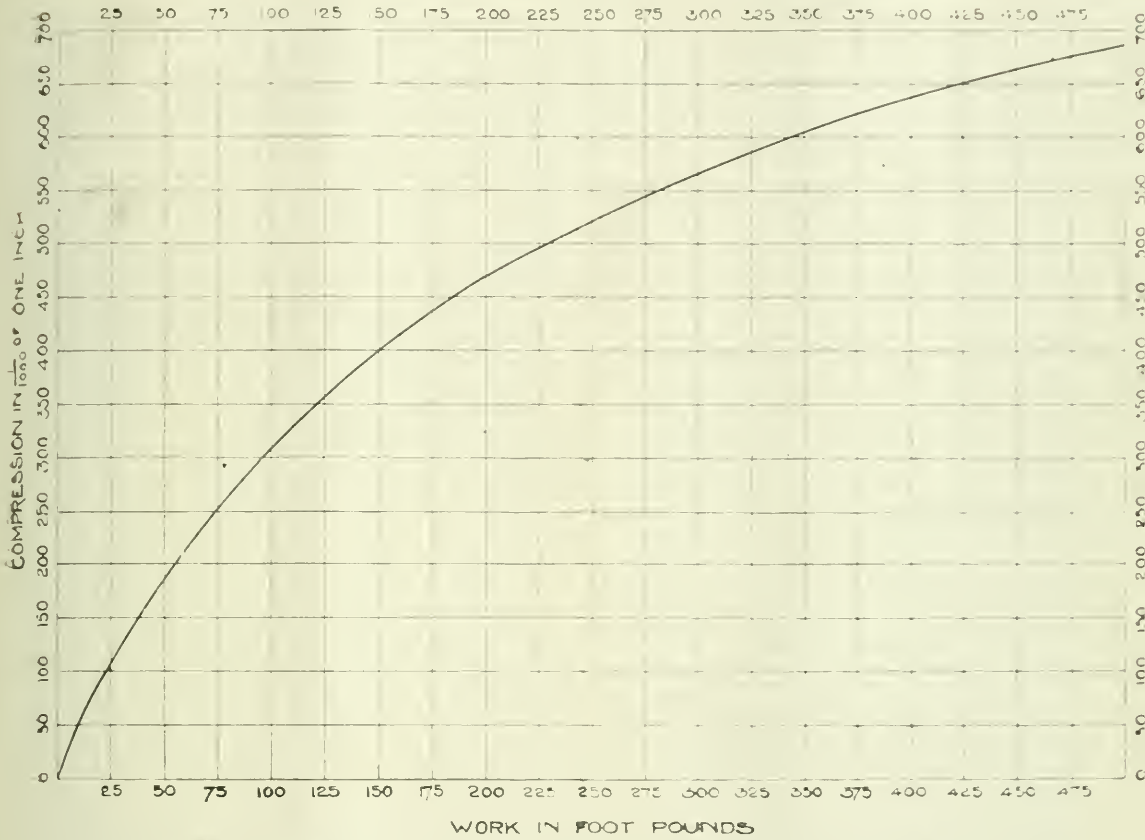
Mr. Whitby is right in answering to Prof. Wilkinson that I did not enumerate paints in my paper, because it would have been tedious to mention all the work carried on the laboratory. Were some of our reports on paints published, they would open the eyes of engineers on these fields. One that I can call to mind, when analysed, was found capable of being made up here for about 10s. per unit, whilst the agent sold it at about 24 times this amount. This profit represents the value of a high sounding name, a drum and an engaging personality or persuasive tongue.

With Mr. Whitby I agree that the viscosity test is improperly reported, hence my remarks. That some better method should be employed is undoubted, and at the present time a committee of the American Society for testing materials is engaged in experimenting on the question. To be of value, the standard must be uniform, and I would draw Mr. Whitby's attention to the above society, which is doing so much in this direction. Similar conclusions also apply to the examination of greases, though it is difficult to conceive of any but comparative tests when experimenting with rope greases, since their cohesive ability in actual work is a very important factor.

Mr. Richardson's remarks are very much to the point, and worthy of thorough investigation. It seems to me, however, that such testing can best be done in actual work—it is rather beyond the scope of the testing laboratory. His remarks probably hit at the reason for the quantity of explosives used per ton of rock broken on these fields, as compared with that used on mines in other parts of the world. The use of a more powerful detonator and the adoption of careful tamping would, I am convinced, considerably reduce this ratio. The value of the explosion apparatus or crusher gauge is undoubted, since it enables one to keep a careful watch on the quality of explosives supplied. It must be remembered, however, that the results obtained are only comparative when the lead cylinders are uniform in quality, and when similar classes of explosives are used. To obtain this uniformity in lead cylinders a large melt is made and a number of cylinders are cast at one time, the lead used being purest sheet lead. The Quinan curve is useful in that it converts the amount of compression into foot pounds of energy, this amount of compression being in itself variable since the more the compression the less is the compressibility.

Through the courtesy of Lieut. Quinan, I am able to reproduce here the curve plotted by him so as to convert the compression into foot-pounds of energy. The quantity of explosive used is 24 gr., or 1·555 gm. A trial is first made with the detonator alone to find the compression due to this. A No. 3 detonator is a convenient size to use. In the diagram the abscissae represent the foot-pounds of work generated and the ordinates, the compression in thousands of 1 in. The diameter of the lead cylinder before use is 1 in., and the length also approximately 1 in. A machine is used for compressing these cast cylinders into this size, any excess being squeezed through two little holes in the sides.

Mr. Caldecott's contribution is valuable in that it tells of a personal experience of the work being done in this direction in other parts of the world. He shows there the slow but steady growth of the largest testing laboratory and, moreover, the recognition of its utility. On railroads where lives are so much at stake it is imperative that only reliable goods must be used, an important evidence of this being in the increased attention now given to the manufacture and testing of steel rails. Why, then, on these mines should



Foot Pounds Machine.—Diagram showing relation of Compression of Lead Cylinders to work done measured as foot pounds.

not every care be taken to see that the best materials only are obtained? The inability to trace accidents to defective material should not militate against constant supervision in testing, but should rather encourage caution amongst managers, engineers, and other heads of departments.

The sole idea of testing materials is to obtain uniformity in quality and provide against fraud. The specifications when drawn up should be such that no impossible clauses shall be included. Each requirement must be legitimate and exact, whilst there should be nothing underhand or unfair in the tests conducted on behalf of the buyer. Our object is to enable the seller to understand the methods employed and to meet him and discuss with him any differences arising in the quality of the supply.

In again thanking you for your kind encouragement, I can only reiterate the feeling of the synoptical nature of my paper, and apologise for the rather summary manner in which I have dealt with various subjects of tests. I feel, like Prof. Wilkinson, that many of them could do with a separate paper for themselves. Should any of the members of the Society find further interest in these remarks, I shall be only too pleased to see them at the laboratory and there exhibit and discuss the testing of all materials in use on these mines.

The Chairman: The original paper was a very good one, and the discussion has caused the author to give us this still better reply. In thanking him I should like to express the pleasure of the meeting at seeing him back again after his serious illness.

THE INCIDENCE OF METHODS OF PAYMENT ON THE EFFICENCY OF MINERS.

(Read at November Meeting, 1907.)

By Kenneth Austin, M.Am.I.M.E.

DISCUSSION.

Mr. W. Taylor: The author states at the commencement of his paper, that there is "a considerable difference amongst workmen even when subject to the same conditions." This no one will dispute, but, that the vast differences in the amount of rock broken by various miners are entirely due to their efficiency or otherwise, I am not prepared to admit.

In hand stoping, if a contractor has a gang of boys, a fair proportion of whom can drill the requisite number of inches, his fathomage depends on the skill, with which he plans his holes, and his monthly cheque on his accuracy in estimating the minimum amount of explosives necessary to break the burden on each hole.

With drilling machines, however, less account is taken of the formation, and the fathomage is more dependent on the number of holes drilled per machine shift. Before the strike, contract prices were based on four holes per machine, one man in charge of two machines. Now, however, one man has to superintend three machines or more, and each machine must drill five holes per shift, if the contractor wishes to come out. The general conditions remain the same, and these conditions are not such as to enable a miner to put in his five holes regularly, as they cause a large waste of time nearly every day. Given a good machine, and good air pressure, I find 1·75 in. per minute the average rate of drilling, although I have timed machines drilling at rates varying from 0·47 up to 2·4 in. per minute. Taking the average, the actual drilling time for a 6-ft. hole is 40 minutes, therefore, with eight hours at his disposal, assuming 9½ hours for a full shift and allowing 1½ hours for getting to his working face, charging up, etc., a contractor could put in 12 holes if there were no stoppages of any kind. Now, of course, it is obvious that a machine cannot be drilling constantly, but it is also obvious that there must be a considerable waste of time if it takes eight hours to do two hours and 40 minutes' drilling. A fair time allowance for work, incidental to running a machine is, I think, the following: three-quarters of an hour for rigging up and oiling the machine, five minutes for changing drills and ten minutes for shifting the machine on the bar to start a new hole; this would give an average of a little over an hour per hole, excluding rigging up, or time for six holes per shift and two rigs. I believe it is the exception to find this amount of work done by miners in charge of several machines, and there are numerous cases where even four holes are not drilled with unfailing regularity.

There are many causes for this waste of time, most of them avoidable, but some of them are: low air pressure (no air at all on occasions); air mains inconveniently situated; leaky hoses; machines out of order; bolts or nuts worn; bars an unsuitable length; incapable boys; broken rock not cleared from the faces; insufficient drill steel; drills that do not follow; drills arriving late underground.

Without a doubt, the cause of most of the trouble underground is the drill question. Nearly all the miners who appeared before the Commission, which has recently sent in its report, stated they were kept short of steel, but still the majority of mines go on running in the same hand-to-mouth style. Drills come down *en masse*, after all the workers are below ground, and tumbled out on the station, where the piccanins have to sort them out, examining the number on each one before distributing them to the different machine runners. There are many obstacles to hinder the quick delivery, such as loose rock on the line, trucks in the way, waiting for a trolley, etc. The pushing contractor sends his machine boys out to bring in their starters, but 10 minutes wasted means 18 in. that might have been drilled, and when a start has been made it is as likely as not the following drills will not "go" without altering the position of the machine. To obviate this loss of time I would suggest that machine steel be sent down every shift to be used on the following shift, in charge of a white man with sufficient boys under him to sort them out into proper lengths and place them, clamped together in sets, in a convenient position. This would necessitate having more drills in use at a time, but also in the ultimate saving of steel, and by the saving of time, reduction in costs.

Machines out of order is the next most fruitful cause of wasted time. Any breakage generally involves the loss of at least one shift, and any sort of repairs, requiring the machine to be taken to surface means two or three shifts lost. If a competent fitter were employed exclusively underground, and had a few spare machines in his charge, wherewith to replace immediately those sent up, the saving in time would pay for his wages many times over. He should inspect all machines at frequent intervals, report on their condition, and be generally responsible for them. The other points require little comment, but if the management and staff were a little keener on removing causes of delay and made every effort to have everything necessary on the spot ahead of time, drilling operations would run smoother and it would be possible for each man to drill an equal number of holes per machine shift.

When all possible causes for wasting time have been removed by the management it would appear that nothing further should be necessary than to lay down the minimum amount of work required per shift and average amount of intelligence in the position and direction of the holes drilled to secure a reasonable fathomage, and there would seem no reason why this should not be achieved by paying a fair day's wage in the same manner as is done for work on the surface. But there is the necessary skill in the use of explosives to be reckoned with and also the fact that a man can always do a little more work if his pocket will be benefited, so that one always comes back to the conclusion that the contract system yields the best results. If stopes were put up to open tender, and all miners were treated impartially, I believe we should get more uniform work and see a still further reduction in mining costs.

REPLY TO DISCUSSION.

Mr. H. Kenneth Austin: The chief difficulty in preparing the original notes submitted was that they might become unsuitable for a technical society, and that they might be considered too sociological, too political. The discussion, however, though small, has not been unimportant. It has been ascertained and established that the individual miner works best and to the utmost of his powers when subject to the conditions of a free, equitable contract, simple, subject to the common law, and based upon fathomage of ground broken.

Much attention has been given to the performances of "show-rock-drills" lately, but little or none has been paid to the human factor, than which there is nothing more important. The number of inches drilled per minute by machines has been noted to two places of decimals, but this kind of thing does not give us with exactness how to attract and cultivate the greater intelligence needed to use them properly, the absence of which we all deplore. The chief advantage of the proposed "free" contract system over long periods lies in the fact that it appeals to the business side of both employer and employed; where Kafir labour has to be used, it is specially advantageous. It is the system which can be most easily adjusted to the constantly changing conditions of the labour market, and the variation in efficiency between one contractor and another. I cannot join with Mr. A. Richardson in deploring the impossibility of finding a "perfect" system, for the simple reason that I consider the "free" contract system is the most perfect one to suit conditions in this Colony.

Mr. W. Taylor has very ably pointed out the practical defects in the details of working, but all the complaints made by him are due to the fact that everything is cut down to reduce the charges for delivering material underground. No doubt, very many of the troubles of contractors are caused by defective organisation in the distribution of material, but this defect bears equally upon everyone. Providing that waste and expense can be prevented, there is no objection to Mr. Taylor's suggestions about the employment of a competent underground fitter to look after machines, but I have clearly laid down that the better plan will be to train the miners themselves; in other words, to attract and to train a better class of man in the elements of his business, before he goes underground, instead of leaving him to fumble about in the darkness of a mine, wasting time and capital in his attempt to "making dirt for the mill" with machines in bad order.

The alleged prevailing conditions, which Mr. Richardson described, where contractors demanding "a price per foot" *for sliding down stopes* appears to be one more fitted for the region of "comic opera" than serious business, and it reflects upon the absurd discipline, if such description applies to any mine on these fields. As Mr. Richardson stated, there is, no doubt, a good deal of "miners' bluff" in the business; there is also no doubt, that the careless and casual method of pricing contracts is the cause of the trouble; the fear of a "cut" or reduction in price causes some amount of anxiety, but the "free contract" system and open tendering with long periods of agreement advocated by me meets this defect.

In summing up the small amount of discussion on this paper, I have to draw special attention to the valuable contribution of Mr. W. Taylor, which is a very correct description in detail of what takes place in underground operations. The attention of those, who are responsible for carrying on underground work is directed to the very practical suggestions made by him. The radical defects in our present system can only be met by the attraction of a better class of trained workman, a skilled artisan instead of the navvy being wanted underground, and when such men are treated with absolute impartiality by capable, even tempered and judicious overseers, there can be no doubt whatever that managers' difficulties with regard to underground work will diminish, and through greater efficiency a further reduction in mining costs will be made possible. It is the common practice to push all responsibility for obtaining better work upon the shoulders of the "shift boss" and mine captain, but, as stated above, a skilled artisan must take the place of the ignorant, careless, bluffing miner, and in order to attract him, the writer has maintained that the incidence of payment for work done has a most important bearing upon whether we get and keep the right sort of men, or not. Progress is to be made by the choice, the conscious choice, of the right path; not by fumbling about in all directions, nor by the indolence which rests on unquestioned, established systems. Content in the correct choice of system, content in what has been secured by methods which work with the most practical and satisfactory results above ground, one may look forward to a time when the underground worker will take his proper position in the industrial world and be paid according to his ability. The payment of the worker underground is necessarily a business matter; there is no reward for him other than the pecuniary one. His work is very exhausting; he starts early from home and returns late, and the rewards should be high enough to enable him to save and establish himself and his family on the spot, free from care and want. Reckless waste, extravagance, dissolute and poverty-stricken lives have frequently been the order of the day amongst miners in the past, indirectly making for high mining costs. Economy, frugality, self-control, enrichment of life, is to be the order of the day in the future. To attain these benefits in a mining community we must establish a better type, a better ideal, and in the attainment of this broadened, fuller, more perfect life and conditions, the method of payment will have a marked, a most material, effect upon efficiency.

The Chairman: I am sure you have all appreciated Mr. Austin's reply, which has sometimes attained to flights of eloquence to which we are not accustomed in this Society, and yet I think it will be found on looking into it that the matter is as good as the manner. Although in some departments of gold-winning we may be able to give points to the rest of the world, yet in the underground work it is evident that we have a good deal to learn.

THE ORIGIN OF THE GOLD IN BANKET

(Having reference to Prof. J. W. Gregory's paper on the same subject, read before the Institution of Mining and Metallurgy.)

(*Read at January Meeting, 1908.*)

By J. S. Curtis (Member).

DISCUSSION.

Mr. T. Lane Carter: I have only had time to take down a few notes on Mr. Curtis' paper, which deals with a very big subject. As a matter of fact, a number of people have attempted to describe the origin of gold in the banket, and they have come to very different conclusions. It is a question which should be tackled by our Geological Survey, and I trust that, in the days to come when the Geological Survey has got through discussing coal and granite formations and so on, they will try to find out for us how the gold got into the banket.

All of you who are acquainted with the work done by the Geological Survey in the United States know what able papers have been given on such auriferous districts as the Rand. I might mention the paper published some years ago on the Cripple Creek district, and one published still further back on Leadville, Colorado. These are monographs, which are authoritative on these particular districts, and I hope we may have the same experience here, and that the Geological Survey will yet give us its final word on this important subject.

There is one man of all others who I wish could have come here and discussed this question, and that is the man who might almost be called the father of the modern science of ore deposits. I mean Prof. Posepny, of Bohemia. In his classical work on Ore Deposits* he mentions the Rand, but unfortunately, on account of the enormous amount of work he had to do in Europe, he never got out to this country, but I think he could have spoken with the greatest authority if he had come here and made a long study of this question. I will read you what he said under the heading of South Africa. Unfortunately he only had a few specimens, and he bases his opinion of the formation here on these specimens and the literature on the subject which had been published up to the time of his death. "In South Africa, at Witwatersrand, in the Transvaal, ancient detrital deposits have yielded a considerable gold production. According to E. Cohen, the Witwatersrand consists of sandstones (which closely resemble that of Table Mountain, at the Cape of Good Hope) and dolomites of high age—undoubtedly palæozoic. Conglomerates of the same age, intercalated among these strata, occur in the vicinity of Johannesburg in several nearly parallel outcrops, and are for certain distances tolerably rich in gold. They are composed mostly of quartz pebbles, sometimes with fragments not entirely rounded, which are united by strong, ferruginous, arkose-like cement. The quartz pebbles are sometimes porous and impregnated with hydrated ferric oxide, thus presenting the peculiar corroded appearance so characteristic of auriferous quartz. The gold occurs chiefly in the cement, immediately next to the pebbles. It is mostly coarse-grained, and sometimes even crystalline. The latter circumstance has raised the question whether the gold has not here been chemically precipitated, and hence whether these are detritus deposits at all. My standpoint in this discussion is that I do not deny the presence of chemical influences in the detrital deposits, although I have personally not happened upon them. So far as I can judge from the treatises of A. R. Sawyer and Charles A. Alford, and from a specimen of the Witwatersrand conglomerate kindly sent to me by A. H. Halder, it is my opinion that the gold was mechanically brought into the conglomerates from still older auriferous quartz veins occurring in the rocks which form the basis of this palæozoic formation; and since the idea of a later entrance of the gold is excluded by the almost vertical position of the conglomerate beds near Johannesburg, I suppose the gold to have been deposited at the same time as the detritus. The greater part of the gold, as has been said, occurs in the cement. There are no vein-like deposits whatever in the conglomerate; and the only chemical changes which could be presumed are confined to the decomposition of pyrites and the segregation of its contained gold."

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* Ore Deposits, by Prof. P. I. Posepny, published by American Institute of Mining Engineers.

Prof. Posepny's theory of the deposition of gold on the Rand is somewhat like that of Prof. Gregory, which the author refutes. Prof. Gregory claims that the gold has been redissolved and again precipitated, whereas Prof. Posepny simply claims that this is placer gold. The author has given us his ideas very clearly, and they differ entirely from those of Prof. Gregory. We may call the author an ascensionist or lateral-secretionist, I think, so far as Rand deposits are concerned.

We have then these three theories, two of which are alike, and the author's, which is entirely different from the other two. As I said before, I have the greatest admiration for Prof. Posepny's work, but seeing that he only had before him the literature published at that time (which was extremely meagre), and only a few specimens to form his opinion on, I feel I must go with the author. It strikes me that the way he sums up his case covers the facts more satisfactorily than either Professor Gregory or Posepny does.

As regards crystalline gold, I remember in 1897 seeing a very beautiful gold crystal, which was found in the Pioneer mine. I understand that these crystals are fairly common, but that was the first and the last that I have seen. This bears out what the author says. Then, too, we find every now and then remarkable masses of gold on the Rand. It is not correct to call them nuggets. Before the war there was a remarkable one found in the Rose Deep, I believe. It was certainly not a nugget. It was a mass of gold deposited from a solution. At the Ferreira mine, in 1904, another remarkable mass of gold was found, similar to that of the Rose Deep.

The author refutes one remark made by Prof. Gregory in regard to the occurrence of gold in the footwall. I do not see what reason Prof. Gregory had to make this statement as it is incorrect to say that the greater occurrence of gold in the footwall is general, for the gold occurs in the hanging-wall as often as in the footwall.

I also agree with the author when he says that Prof. Gregory's remark with regard to the redistribution of gold after solution is far-fetched and complicates matters, and I do not see that it is necessary at all. Another point that the author refutes is in regard to the gold in the bastard reefs. This is not correct either. Those of you who have examined the country rock and these bastard reefs very carefully, know that gold often occurs in both places in weighable quantities. The bastard reefs do not contain much, but they certainly do contain an appreciable amount. The author refers to the "bullet" pyrites at Rietfontein. I do not know whether he is acquainted with the West Rand, but there is a very interesting occurrence on the Lancaster mine in the Kimberley reef series. As you know, this series is not worked anywhere on the Rand except at the Lancaster and the Lancaster West. Out there it is their main gold producer. In the Kimberley series you often see pellets of pyrites of varying sizes. There seems to be a connection between the occurrence of pellets and the amount of gold in the rock.

Another point which the author brings out is that gold jumps from one reef to another. A most interesting point on the Rand is that a reef which is of very good grade in the central Rand increases or diminishes in value, as the case may be, as you go either east or west.

The author mentions also finding several other minerals in the banket. I can bear out this statement. We have very often found interesting collections of minerals in the banket itself. In one place on the West Rand we have quite a large number of faults, and there has been a large amount of water action, and in consequence there is some deposition of different minerals. I have a piece of banket which is full of galena and zinc blende, and of high gold value.

I disagree somewhat with the author when he says that the bigger the pebble the richer the ore. I can show him in my district some of the finest and most superb looking pebbles the world ever saw, and there is no reef in the country which can compare with it in looks, but unfortunately when you assay it you find that the gold is missing.

With regard to the question of chutes versus patches, I agree with the author that we get well defined chutes on the Rand. Besides the Central Rand chutes there are several others, one of which I am well acquainted with, being the Champ d'Or chute. There is one of the best examples on the Nigel mine. These bear out the author's statement where he refutes Prof. Gregory's assertion that chutes do not occur on the Rand. Prof. Gregory says they are patches, but I agree with the author that we get well defined chutes on the Rand.

The Chairman: We all thank Mr. Lane Carter for his enlightening remarks. I should like to make a few remarks myself. In discussions of this sort it would be well if chemists and geologists would consult together, for as a matter of fact I think the two fields are kept too wide apart. A chemical point which has some bearing on this discussion, is the extreme instability of all the compounds of gold even at comparatively low temperatures. There is, as far as I remember, no compound of gold which will stand 300° C., so that if the gold has come up from any depth it is difficult to see how it could have come up as any sort of compound of gold in solution. The alternative supposition to that made by geologists is that the gold might be in simple solution in water. Now it is true that for the purpose of expounding the modern theory of solution it is necessary to assume that all metals are soluble in water, but it has been found that the solubility of gold in water is exceedingly small, possibly the smallest of all the metals, and something like one part in a million millions. The other point is with regard to the physical properties of water. Geologists sometimes give the impression, in talking of other aqueous problems, that water can exist at any temperature. Now the critical temperature of water is about 330° C., a comparatively low temperature, and one which is attained at comparatively shallow depths in the earth, whence it follows that no sort of aqueous solution at all can exist above 330° C. One must therefore assume from that, that if the gold really has come from any great depth, it cannot have come up with water as a solvent; and I therefore think that the theories which make iron pyrites the gold solvent, are far more likely than the others.

EXPERIMENTS IN FIRE ASSAYING AT THE REDJANG LEBONG MINE, SUMATRA.

(Read at September Meeting, 1907.)

By G. B. Hogenraad (Associate).

DISCUSSION.

Mr. J. E. Clennell (contributed): Although I have not had the pleasure of seeing this paper in the September *Journal*, I gather from the discussion on it that it covers the same ground as a report which I received some time since from Mr. Hogenraad, and hope I may be allowed to make a few comments on the points raised, if the paper is still open for discussion.

I entirely agree with Mr. Wilmoth and Mr. Whitby that the use of borax was not the cause of the losses of gold and silver observed; in my opinion, the low results with the original flux were chiefly due to insufficient litharge, or to excess of charcoal, rendering the charge somewhat infusible and hence causing mechanical losses. As this flux had formerly given satisfactory results, I am inclined to think that some change has occurred in the average character of the samples dealt with at the Redjang Lebong assay office, probably an increase in the amount of pyrites and possibly also of selenium. However, the point of chief interest is the influence of borax on the assay charge, and as I have recently had occasion to test this point in reference to another class of ore, the conclusions arrived at may possibly be of some interest to assayers on the Rand.

Without going into details, I may say that the ore on which these tests were made was similar in general character and appearance to the Rand banket, but was practically free from iron pyrites, though carrying considerable quantities of iron in the form of black oxide, perhaps titaniferous, and that a large part of the gold appears to be intimately associated with the oxide of iron.

An average of 297 assays made without using borax gave the result 18·169 dwt. An average of 297 assays made on the same samples, and with similar fluxes, but containing borax, gave the result 18·185 dwt., showing a slight advantage for the borax flux. Unfortunately, I have no opportunity of experimenting on an ore rich in silver, but I think it may safely be concluded that the use of a moderate amount of borax is at any rate not detrimental in the assay of free-milling gold ores.

In the course of these experiments the following points were observed:—

(1) A large excess of borax gives a hard, stony slag, very difficult to separate clean from the lead button. In hammering out, the slag flies to pieces violently, carrying with it a portion of the lead; some part of it is sure to be lost, unless extreme care be taken in collecting all the fragments. This may very possibly be a cause of some of the observed losses through the use of borax.

(2) When no borax at all was added, the button usually separated quite clean from the slag; the latter was amber-coloured and, while cooling, a scum of straw-coloured crystal collected on the surface. The addition of borax rendered the charge distinctly more fusible; this seems to be contrary to Mr. Hogenraad's view, as he apparently ascribes the sticking of the lead to the moulds to the extreme fluidity of the non-borax slag. I should be inclined to draw the opposite conclusion; the fact of the lead leaving the pot before the slag, would seem to show that the latter was not sufficiently fluid.

(3) The non-borax flux was considerably more deliquescent than the ordinary borax flux; this,

however, caused no inconvenience, unless the fusions happened to be left in the moulds for some hours after pouring.

(4) By using a *small* quantity of borax in the charge, not exceeding 5 to 10 gm. per A.T. of ore, it was found that a uniform fusible slag was produced without any noticeable amount of the scum above mentioned: the buttons could be detached from the slag by a slight blow, and none of the lead adhered to the slag.

With regard to the reducing power of charcoal on litharge, Mr. Hogenraad is, no doubt, correct in stating that the button of 34 to 35 gm. lead is obtained by fusing a mixture of 1 gm. charcoal with an excess of litharge, in the absence of interfering substances. But such conditions do not obtain in an ore assay. A part of the charcoal may be consumed in reducing metallic oxides or, in an uncovered crucible, may be oxidised by the atmosphere. On the other had, a part of the litharge may go to form silicate of lead with the quartz of the ore, this compound being only reduced with difficulty by charcoal, and some lead may be produced by the reducing action of metallic sulphides, etc., on the litharge. Hence the weight of lead obtained cannot be predicted from the composition of the flux. In general, an excess of litharge over the theoretical amount should be used. In the case of the ore on which my experiments were made, the average quantities of lead obtained per gramme of charcoal in the charge, with three different fluxes, each containing soda, borax, and a moderate excess of litharge were respectively 19·3, 18·7, and 18·3 gm., this low result being probably due to the oxidising effect of the ferric oxide in the ore.

The fluxes finally adopted as giving, on the whole, the most satisfactory results, were as follows:—

	A	B	C
CLASS OF SAMPLES.	Mine and Mill.	Cyanide Originals and Slimes.	Cyanide Residues.
Flux: Parts by weight:			
Bicarbonate of soda ...	2,500	2,400	3,600
Fused borax glass ...	200	200	400
Litharge	2,500	2,400	3,600
Charcoal	60	50	60
Weight of charge:			
Ore for assay ...	1 A.T	2 A.T	4 A.T
Flux (grammes) ...	120	200	300
Cover of soda (grms.)	10	15	20
Crucibles used (Battersea)	G	H	No. 12
Cupels (Mabor) ...	No. 7	No. 8	No. 8
Approximate weight of lead buttons obtained (grammes)...	30	40	45
Lead reduced per grm. charcoal (grammes)	22	20·2	19.2

(varies slightly with quality of charcoal and nature of ore).

It is perhaps necessary to point out that the charges adopted were partly determined by the stock of fluxes and crucibles available, and could, no doubt, be improved under ideal conditions; for example, carbonate of soda might advantageously be used instead of bicarbonate.

I have purposly omitted all details of my experiments, as these have already been submitted for publication elsewhere.

SOME FEATURES OF SILVER ORE TREATMENT IN MEXICO.

By W. A CALDECOTT, B.A., F.C.S. (Member).

PART II.—THEORETICAL CONSIDERATIONS.

Mr. W. A. Caldecott (contributed): I wish to point out an error which has occurred in the second part of my paper. On p. 267 of the March *Journal*, first column, 10th line, read "*50* c.c. of each of which was titrated with *0·01*% sodium sulphide solution," instead of as there printed.

NOTES ON RAND MINING.

(Read at March Meeting, 1908.)

By TOM JOHNSON (Member).

DISCUSSION.

Mr. A. L. Edwards: The author in his valuable paper puts forward the suggestion that the sand residues from the cyanide works should be hauled to the mine shaft and utilised in filling up the old stopes, claiming that the residues could be hauled to the shaft at a cost considerably below that of hauling the same material to the dump. As far as comparisons of mere costs go, it will at once be conceded that he is right, but the more pertinent question which will appeal to those most conversant with the physiological effect, which the combination of cyanogen and the acid waters of the mine will produce is, "is it safe for the workers down below?" and we must bear in mind that the health and safety of the men underground is of even greater importance than economy of working costs, however desirable this might otherwise be.

It is essential that in our endeavour to reduce working costs we must not be allowed to overlook the health and welfare of the employee, for, anxious as we all are, to reduce working costs to the absolute minimum, and willingly as we would all help to attain this highly desirable end, we must recognise the fact that this is one of the economies that cannot be recommended for adoption, because the health and safety of the men employed in the mine must be the guiding and deciding factor in this particular instance.

It may not be out of place here to point out that this intended method of dealing with the sands residues has already been tried on these fields and abandoned. When the cyanide process was first being adopted here, it was one of the conditions obtaining in the contract between a company and a syndicate that the whole of the treated sands had to be carted to the company's mine and dumped down the old stopes. This dumping continued until the company's manager found that the results that ensued were not conducive to the health of the employees engaged below. It may be information to some, that Blyth in his book on poisons, p. 214, draws attention to the fact that all cyanide workers invariably suffer from giddiness, noises in the ears, difficult respiration, pain over the heart, a feeling of throat constriction, loss of appetite, nausea, etc.

The author in his endeavour to reduce working costs may rest assured that, he will have the whole-hearted support and endorsement of the members of this Society, and it will indeed be a glorious day for the mining industry when the present disproportion between the mining and milling costs shall have been sensibly reduced. The cost of mining and hauling a ton of rock is approximately 10s. per ton to-day, yet a ton of ore can be treated in the reduction works and have 95% of its gold values won at an inclusive cost of 5s. Now the difference between these two amounts is altogether too great to allow of their being accepted as standards of what the future working costs should ultimately be, and it is to the reduction of costs in this direction that we are all earnestly looking forward.

The meeting then closed.

RESCUE APPLIANCES: LESSONS FROM GLENCOE.

(*Read at March Meeting, 1908*).

By H. Kestner.

After a historical introduction in which the author discussed earlier forms of apparatus, he proceeds as follows: The first impulse to construct a circulation rescue apparatus was given by Prof. Albert Habets of Liège. At the Brussels exhibition, there was introduced for the first time, in the respiratory appliances of Messrs. Theodore Schwann and E. Schulz, a principle entirely different, and which appears thoroughly worthy of study as an advance towards the solution of the problem in which we are interested. As my paper would become too extended in reviewing the ideas of Prof. Schwann, I will abstain from describing this special apparatus. However, I would here mention that it has been recognised, that he had actually produced portable oxygen breathing appliances half a century ago. It is especially to be noted that his ideas were not limited to those appliances in which gaseous oxygen under pressure could be carried about, but that he had already fully considered the employment of a material for the simultaneous absorption of carbonic acid and evolution of oxygen, in the form of hydrated barium peroxide, in a similar way to that employed in the pneumatogen of Prof. M. Bamberger and Dr. Fr. Boeck.* It is difficult to understand why the Schwann apparatus had for so many years been entirely forgotten, although it had been twice publicly exhibited, and why the same appears to have been the case with the apparatus of Mr. H. A. Fleuss, which was designed on similar lines. This latter apparatus, exhibited in the year 1880 in Great Britain, was much discussed at the time; it was practically employed in the Seaham Collieries in the county of Durham, and subsequently modified at the German Government mines at Saarbrücken.

The employment of the Schwann apparatus was vehemently opposed by Prof. Paul Bert, who maintained, that the respiration of pure oxygen was injurious to man. Prof. Schwann was able completely to refute this view by a series of exhaustive experiments upon men and animals. The refutation was not, however, very strongly believed in, and the belief in the injurious effects of breathing pure oxygen exists even at the present date, a circumstance upon which I will touch subsequently. One circumstance, which must have hindered, in a great measure, the employment of the Schwann apparatus in general mining practice is, that it did not supply the quantity of oxygen really necessary for men undergoing muscular exertion. Prof. Schwann assumed that the quantity of 30 cub. in. (0·3 litre) of oxygen per minute was sufficient for the wearer of the apparatus. Whether he considered this amount really sufficient for a man engaged in active work, or whether he set the figure so low in order to diminish the weight of the apparatus to be carried by the operator, I am unable to say. It is, however, clear to-day that it is not possible to maintain human breathing satisfactorily with the above quantity of oxygen, when there is violent muscular exertion. This matter has been made clear by the physiological experiments of various investigators, amongst whom I may mention Dr. Leo Zuntz. In his "Investigations upon the supply of air and the conversion of energy in cyclists" (Untersuchungen über den Gaswechsel und Energie-Umsatz des Radfahrers, 1899, Berlin), the consumption of oxygen during the period of rest of the human body is given as 16·1 cub. in. (263 c.c.) per minute, as compared with the value, for rapid movement upon the bicycle, of 143·4 cub. in. (2,331 c.c.) per minute, so that violent exertion requires about 8·9 times as much of this gas as is absorbed when the body is at rest. The speed attained on the bicycle with the above high consumption of oxygen amounted to about 13·294 miles (21·394 km) per hour, which can be looked upon as an ordinary speed for an experienced cyclist.

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* See this *Journal*, vol. vi., p. 197, by A. Prister.

Experiments performed at the Shamrock mines in Westphalia have led to the conclusion that, in a breathing appliance, the regular supply of 122 cub. in. (2 litres) of oxygen per minute is necessary, and that this is amply sufficient for vigorous exertion. These tests do not pretend to have the value of experiments such as might be carried out in a scientifically equipped laboratory by the aid of trained experimenters. Through the above-mentioned experiments of Dr. Leo Zuntz and other investigators, the experiments at the Shamrock mines received a most valuable confirmation, and these experiments laid the foundation for the principles controlling the design of the magazines for oxygen under high pressure for breathing-appliances. The modern breathing-appliance differs in the following notable points from that devised by Prof. Schwann, besides the above noted modification in the quantity of gas :

1. Valves have been done away with and replaced by the automatic circulation of the gases through the regenerating vessel without making any demands upon the respiratory muscles ; and by bringing the fresh gases direct to the mouth of the bearer of the apparatus.

2. The mechanical purification of the regenerated gas, together with its simultaneous cooling and removal of moisture, by a porous absorbent placed close behind the connecting pipe in the regenerating vessel, the pipe being placed behind the regenerating vessel.

3. The automatic unloading of the air bag containing respirable air, carried on the wearer's breast, by means of a spring escape valve.

4. Closing the nose by application of the smallest possible surfaces, namely, by plugging the nostrils.

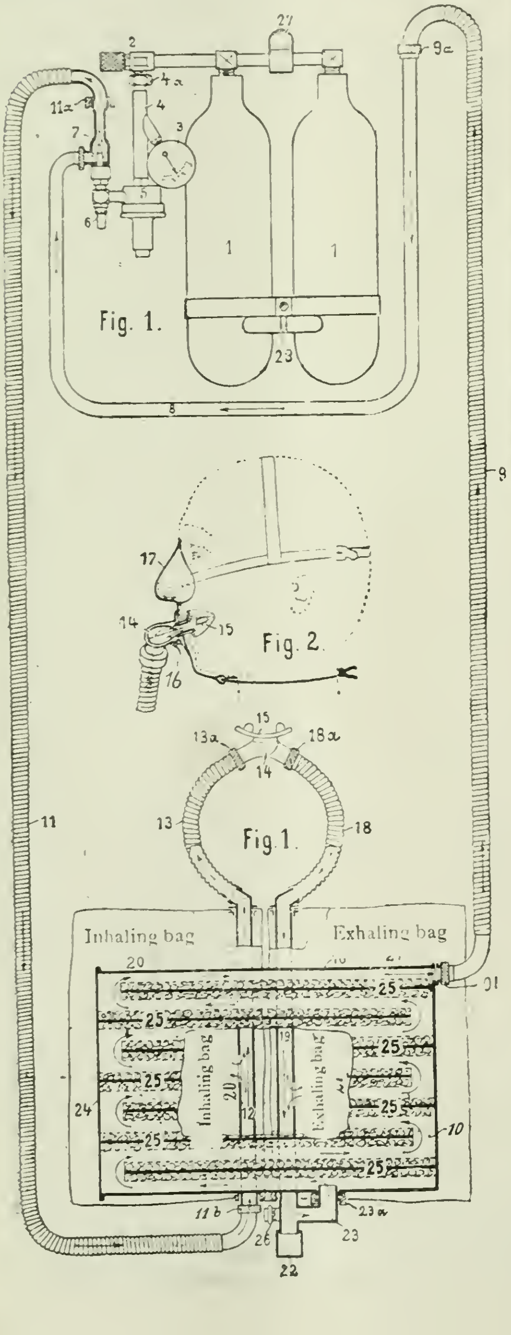
5. By adapting the outward form of the entire appliance to the narrow space generally available in accidents of the kind here referred to, and the protection of the apparatus against external injury.

How these ideas have been worked out will be shown by the construction of the "Shamrock" apparatus, which, I may say, was exclusively employed by the rescue party in their work at the Courrières mines. The idea of doing away with the valves was already worked out in the year 1896, in the pneumatophore, which was brought out by Messrs. Rudolf Walcher von Uysdale and Prof. Gustav Gartner.

I do not propose to enter upon the question of valve defects, which have been exhaustively referred to in papers upon the subject. Various engineers have, at different times, arrived at the conclusion that the power contained in the highly compressed oxygen is sufficient to cause the expired air to traverse the vessel, wherein it is freed from carbon dioxide. This object has here been attained by injector 7, delineated in the diagram. The oxygen in the two bottles is compressed to 1,764 lb. per sq. in. (120 atm.)

The "Shamrock" rescue apparatus exhibits very considerable improvements as compared with other existing constructions. In particular the so-called automatic device and the injector have been improved, so that the apparatus is now capable of yielding a guaranteed circulation up to 60 litres (about 15 gallons) of purified air per minute. The mouth jet of this apparatus is unique, since every possibility of dead space is avoided. The purified air, rich in oxygen, is introduced directly into the mouth of the wearer of the apparatus, while the exhaled breath is also sucked off at that point, so that no exhaled air can be inhaled again. A very noticeable and very appreciable advantage lies in the fact that the usual nose clip is done away with, the use of which had the disadvantage that it could be knocked off, thus entailing great danger to the user. The clip is replaced by a nose cap, which not only securely retains the plugs of cotton which, greased with vaseline or the like, are inserted into the nostrils, but also makes it impossible for the mouthpiece to drop out of the mouth. The mounting of the regenerator has been retained on the chest and as near as possible to the mouth in view of the great advantages of this arrangement. The construction of the regenerator, however, has been completely altered, and it ensures a more perfect absorption of the carbonic acid than has ever been attained before. The regenerator is so arranged that it can be easily changed. The same is manufactured in two different types. In the one type the regenerator may always be refilled, while in the other type the complete regenerator is thrown away after use. The air bags, which act as reservoirs, are mounted between the regenerator and the chest, so as to protect them and also to prevent their being compressed. They are of such a size, and are connected to the circulating current of air in such a manner, that even in the event of the hardest work being done, a sufficient supply of fresh air is available, and they also contain sufficient room to permit of comfortable exhalation.

The mode of operation of the apparatus is briefly as follows:—After the check valve 2 has been opened, the compressed oxygen, contained in the two steel bottles 1, 1, passes through the T-pieces 4, which is provided with a pressure gauge 3, to the reducing valve 5, which is provided with a safety valve 6. The pressure of the oxygen is hereby reduced to about 1 atm., and it then enters into injector 7. The injector, by means of the high velocity of the flowing oxygen, sucks the air (purified from carbonic acid) coming from the regenerator 10 through the pipe 8 and hose 9. The purified air is then mixed in the injector with fresh oxygen and is pressed by the same through the connecting hoses and tubes 11, 12, 13 and the mouth jet 14 into the mouth of the wearer as good, wholesome air, rich in oxygen and purified from carbonic acid.



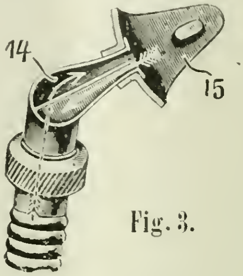


Fig. 3.

The mouth jet 14 is divided into two chambers by a partition wall, as shown in figure 2. Through the upper chamber the purified air, enriched in oxygen, is forced directly into the mouth of the wearer of the apparatus, while the exhaled air is sucked off through the lower chamber. Thus, in this arrangement, any possibility of dead gas space is avoided. Any saliva which might be formed, flows through the tubing 18 and the pipe 19 directly into the saliva trap 22, which is closed with a screw cap. The nose of the wearer is closed by plugs of cotton, which are inserted into the nostrils and which are greased with vaseline or the like. They are prevented from dropping out by the nose cap 17, so that a perfect and comfortable means of closing the nose against the access of the surrounding noxious gases is afforded, the mouth jet 14 being at the same time securely retained with the mouthpiece 15 by the strap 16. The exhaled air is carried off directly from the mouth, as already mentioned, by the mouth jet 14, hose 18 and tube 19, and is admitted into the regenerator 10. The exhaled air passes through the elbow pipe 23 into the lower part of the regenerator 10, through which it flows in the direction indicated by the arrows, being then sucked off at 9b. As shown by the drawing, the regenerating substance, caustic potash and kieselguhr, is arranged in several layers 25, 25, in the regenerator 10. These layers consist of wire cages, there being this great advantage in their construction, that both the upper and the lower surface of the carriers of the regenerating substance are rendered available. In view of the comparatively large surface created thereby an exceptionally good absorption of carbonic acid is effected. In the middle of the several boxes 25, 25, an impervious mass of absorbing substance is arranged, which prevents the air to be purified from passing through the regenerator in any other but the predetermined manner and absorbs at the same time any moisture that might be formed. The regenerator is furnished either completely filled and is thrown away after use, or it is provided with a removable cover 24 and may be refilled at the mine by the insertion of a new charge 25, 25, so as to become again available for service. This arrangement possesses the advantage that it is possible to control the condition and quality of the regenerating substance and by this means accidents are avoided. The regenerator is easily exchangeable.

The pipes 12 and 19 pass through the inhaling bag 20 and through the exhaling bag 21 respectively, and are in communication with these two bags by suitable openings. The inhaling bag 20 serves as a storage receptacle for the purified air enriched with the oxygen, in order to be able to dispose of a sufficiently large supply for inhaling purposes when an extraordinarily large amount of air is needed. The exhaling bag 21 serves as a compensator when quicker exhaling takes place.

Inasmuch as the apparatus furnishes considerably more purified air, enriched with oxygen, than is required under ordinary operating conditions, the lower end of the pipe 19 is provided with a blowing-off valve 26, through which the excess can escape. In order to keep the apparatus always ready for use, the regenerator must be protected against the entrance of atmospheric air, while it is not in use, arrangements being provided for this.

The contents of the two bottles of oxygen are sufficient for fully two hours' work, pressure gauge 3 indicating exactly the number of minutes the oxygen contained in the bottles will suffice for. The hand of the pressure gauge must point to 120 after charging.

The construction of the apparatus is such that the different parts are distributed both upon the chest and back. The part which rests upon the back is completely protected, if desired, by a flat knapsack, and as there are no projecting parts the possibility of it getting caught and being injured is almost impossible. The wearer of the apparatus is not hindered in any way in his movements and is able to enter with the apparatus into the most difficult and narrowest places of the mine. The covered opening provided on the knapsack permits of the pressure gauge being observed.

The range of vision is not interfered with in any way in this apparatus, smoke goggles being needed only when inconvenience or injury to the eyes, by fumes or the like, is to be feared: in most cases it is impossible to work without goggles.

The same apparatus may be used in connection with the helmet, and in this case the two hoses 13 and 18 are screwed directly on to the helmet.

As regards the analysis of the samples of air taken in certain experiments with the Shamrock apparatus, I wish only to refer to those which were taken from the tube leading to the mouth and close to the mouth during the period of inhalation of the wearer. These analyses have been published in "Die jungste Entwicklung der Atmungsapparate unter besonderer Berucksichtigung der auf der Zeche 'Shamrock' I II neuerdings ausgeführten Versuche," by Mr. O. A. Meyer, in *Glückauf*, 1904, vol xl., p. 1123. Seven samples were taken, and the average analysis showed :—

Carbon dioxide... 3·87 % Oxygen ... 31·3 %

After an uninterrupted period of one hour seven minutes 3·2% of carbon dioxide was recorded, being notably beneath the above-named average contents. Thirteen minutes later, however, the carbon dioxide rose to 3·6 %, whilst an hour and 37 minutes after the commencement of the exercises, the carbon dioxide had risen to 7 %. Although these results were not in every respect satisfactory, they, nevertheless, showed a notable improvement over the injectorless pneumatophore.

The result of scientific investigation, that the supply of an excess of oxygen would not do away with the evil consequences of a simultaneous excess of carbon dioxide, was of the greatest importance in the design of the breathing appliances and in the regulation of its mode of working.

It was recognised that it was necessary to construct the apparatus so that the exhaled air should be removed as easily and as rapidly as possible from the mouth and delivered to the regenerator, whilst fresh air should take its place. These investigations led to the construction of the mouthpiece as shown on the diagram, from which you can see that there are two different passages, one for the inhaled air and one for the exhaled air. After this modification had been made, the composition of the gaseous mixtures present within the apparatus was again determined by the analysis of numerous samples of gas. The results showed that the desired object was attained.

The regenerator should always be freshly charged with caustic potash before starting exploring work. The regenerator charges are packed and kept in air-tight closed tins. It has been considered advisable to prepare the charges in the form of cartridges. The mechanic in charge of the rescue apparatus should protect his hands from the action of the caustic potash by wearing india-rubber gloves when charging the regenerator.

The fact that the charging of an apparatus of the Shamrock type can be performed upon the mine is of the greatest importance, and the charging of the regenerator requires only a few seconds. A charged apparatus of the Shamrock type has been kept for about six months at the Shamrock mine, and analyses have shown that the caustic alkali had lost practically none of its power for absorbing carbon dioxide.

In the lower portion of the absorbing regenerator a certain quantity of kieselguhr is charged, the object of which is to absorb the moisture condensed from the exhaled air. The amount of moisture given off from the human breath may be calculated for a period of two hours at about 3·77 oz. (107 grammes). In addition to this quantity, an amount of saliva, differing widely with differently constituted people, has to be added, so that it may be certainly assumed that about 4 oz. of water have been given off by the wearer of the apparatus. As stated already in the description of the apparatus, the saliva is collected in the collecting vessel No. 22. Furthermore, the system of hoses and tubes already described presents the valuable subsidiary effect, that the gas, which is considerably warmed by the action of regeneration, is thoroughly cooled. The above-named tubes can easily be cleaned by unscrewing the stoppers.

A small piece of apparatus is combined with the breathing sack, which, although apparently very unimportant, is nevertheless indispensable. It is the relief escape valve No. 26 on the diagram. It has already been proved that the consumption of oxygen varies very widely in accordance with the muscular activity of the human body. If the appliance delivers a uniform quantity of oxygen, periods of deficiency and of excess of oxygen must ensue, as was foreseen in the construction of the Schwann appliance.

If the quantity of oxygen supplied is as abundant as it is in the Shamrock type, the presence of an excess of oxygen will naturally occur at those periods when the physical powers of the wearer of the apparatus have the least demands made upon them. At such times, so great a pressure is produced in the breathing bag and the tube leading to the mouth that the power of the lung muscles no longer suffices to discharge the air after respiration. The consequence of this is that gas, rich in carbon dioxide, accumulates in the lungs, and this might lead to asphyxiation by carbon dioxide. In order to avoid this, the blow-off or relief valve No. 26 is provided. It is an easily movable spring valve.

The leather nose cap has no relation to the exclusion of the air, but, on the other hand, it fulfils the very important object that it prevents the mouthpiece from dropping out. It is, therefore, impossible that sneezing or coughing of the bearer can cause the mouthpiece to drop out of his mouth, and at the same time he has not the feeling of being shut in, as occurs with helmet appliances, which is apt to make him nervous in critical moments. By distorting and opening his mouth very widely, he can also make direct communication with the external air. This is an immense advantage, for one has to deal in this matter so largely with questions of personal sensation.

A notable speciality of the Shamrock breathing apparatus is that it has been adapted in all its parts to the conditions existing in a mine damaged by accident, and it is the result of practical experiments. From various places the management of the Shamrock mine has received communications, giving the experience derived from various types of breathing appliances, under the most varying conditions of mining, and all the material so collected was employed for the further development of the apparatus. It is in this way that the external shape of the Shamrock has been derived.

The apparatus now in use at the Shamrock colliery, has a small electrical contact, which is in communication with an electric bell and the pressure gauge. As soon as more than half the oxygen is exhausted, the wearer is reminded by the bell sounding at intervals of five minutes that he must think of making his way out the danger zone. This warning arrangement makes the wearer easier in his mind.

A few words may be added about the protection of the eyes. Only in exceptional cases as, for instance, in fighting fire in a mine is it necessary. If, however, this exceptional case should arise, goggles are by far preferable to a helmet. It is possible to take the goggles off without interfering in the least with the breathing apparatus, and so wipe off perspiration, which accumulates in the angles of the eyes ; this cannot be done when the face is covered with a helmet.

It was a strange coincidence that the first Shamrock apparatus in this country arrived about a fortnight before the Glencoe disaster occurred. About three weeks ago I went down to Glencoe with the "Shamrock" and the reviving apparatus, and I have pleasure in relating my experience there.

When I arrived at the mine and unpacked my apparatus, the apparently intricate construction created a somewhat unfavourable impression, but

I at last persuaded the mine captain to use it, and I can assure you, that he is at present a staunch believer in the Shamrock. At Glencoe I made the acquaintance of the Vajen-Bader helmet, which was tried there for rescue work. This helmet is, in my opinion, quite unsuitable for coal mines or for any kind of underground work in connection with serious accidents. The helmet is a leather cover, which the wearer puts over his head, and which is fastened to the body by means of straps. At the back of the helmet there is an air cylinder with pressure gauge. This air cylinder has to be filled on the surface with fresh air by means of an ordinary bicycle pump, and the air is compressed to about 25 lb. per square inch. The wearer of the apparatus is able to regulate the inflow of the air into the helmet by means of a small valve which he can easily reach. It is supposed that the air cylinder contains sufficient air for one hour, but I do not believe that there is sufficient air for half an hour, when there is violent muscular exertion. Another great drawback in connection with the use of this helmet in mining rescue work, is that the wearer is compelled to breathe over and over again the already respired air. You will find that the miner in rescue work will prefer a breathing apparatus without a helmet.

The lesson arising from Glencoe for the different Governments and collieries of South Africa, is the immediate creation of central rescue stations in the different coal districts. I believe that if an exploring party had been equipped with rescue apparatus they could have gone forward immediately after the third explosion and saved several of the valuable lives which were lost by the effects of the after damp.

The finest rescue station in existence is at the Shamrock mines, in Westphalia. The Shamrock mines have installed in a special room ten Shamrock appliances of the mouth breathing type, and one of the helmet type. The latter is kept so that the rescue parties may be perfectly familiar with this form of the apparatus, and, in case of need, may be able to use it and render help at some mine where only helmet appliances are kept in store.

I now beg to submit to you my views regarding rescue stations for coal mines in South Africa. Each rescue station should be provided with the following appliances :—

Five circulation appliances ; 2 oxygen reviving appliances ; 5 to 10 safety lamps, of the electric dry battery and accumulator type ; 2 or 3 light stretchers.

To these appliances should be added :—

(*a*) 1 or 2 steel bottles of large size, filled with oxygen ; (*b*) the necessary pressure pump to transfer the oxygen from the large bottles into the smaller ones of the rescue apparatus ; (*c*) smoke goggles, or helmets, for use in smoky atmospheres ; (*d*) all necessary spares, such as pipe connections, spare hoses, spare mouthpieces, and especially spare rubber joint rings.

At the Shamrock mines a notice is posted in the rescue room, reading as follows :—

"Before using, test the pressure of oxygen by means of the pressure gauge, also the suction of the jet at the water gauge. Do not forget your electric lamp. Enter the mine only in parties of four men and a leader."

The leader is not allowed to work ; his duty is only to control the men, to give them signals, and to carry in his hand an oxygen reviving-apparatus.

Though attempts may be made to produce artificial respiration in the well-known manner, by moving the arms, etc., of the patient, this will fail to accomplish the desired results in most cases. First of all, a person who is well experienced in this work is needed, and in the second, this artificial breathing by movement of the limbs of the patient, is absolutely out of the question in cases where the patient has also received such injuries as fracture of the arm or the like.

An apparatus, which in similar cases is always ready for use, and which is very well adapted for the purpose, is the oxygen reviving apparatus (patent of Dr. Brat). The operation of the apparatus is absolutely reliable, and the manipulation is simple, so that anybody can use the apparatus without difficulty, the emptying of the lungs and the exhalation taking place by the production of a kind of vacuum by means of an ejector, which acts upon the lungs.

It would be as well to have a transportable telephone apparatus in the rescue station, ready to be taken down the mine in cases of serious accidents. This apparatus could be carried by the rescue party through the gas-laden atmosphere and communication with the surface maintained. Everyone, I am sure, will acknowledge the great advantages of such an arrangement ; it enables the manager, after he has seen what the position of affairs is, to keep strict control of the exploration of the mine. Rescue appliances have a real value only when the men are trained in the handling and use thereof.

Whenever any defect is discovered in the apparatus, due to an unsufficient supply of oxygen, excessive heat in the regenerated air, or any other cause, the wearer must, without delay, return to the shaft or safety station, accompanied by one fellow explorer. It is better to be sure than sorry. I think it is also absolutely necessary to keep one reviving apparatus underground, in case of unexpected gassing.

Proceedings

AT

Special General Meeting, April 18, 1908.

Prior to the ordinary meeting a special meeting of members was held for the purpose of considering, and if approved, adopting with or without amendment, the following proposal brought forward by the Council:—

The Council, having had under consideration the annual cost to the Society of the *Journal* and the general expenses of administration, etc., per member,* finds that Associate Members do not bear their fair proportion of the cost, and recommends:—

(*a*) That on and after the 1st day of July, 1908, the annual subscription for all Associates be increased from One Guinea (£1 1s.) per annum to One and a Half Guineas (£1 11s. 6d.) per annum; and

(*b*) That the necessary steps be taken to amend the Constitution and Rules accordingly.

The Chairman (Dr. J. Moir): You have heard the resolution read, and I have much pleasure in moving that it be adopted.

Prof. J. A. Wilkinson seconded.

Mr. W. Taylor: There is just one point that occurs to me, though I suppose that it has already been carefully considered. I know of one man—and one only—who says that if the Associate subscription is raised he will not continue to be one any longer. The reason is, that he does not take much interest in the Society, but he likes to read the *Journal*, and he thinks that is worth the guinea, but if the subscription is raised to one and a half guineas he will not think it worth his while to continue his connection with the Society.

The Chairman: I think there must be others in the same frame of mind. Of course, our attitude as a Council is that we cannot run the Society as a charitable institution, and so long as we are five or six shillings short of the actual cost of producing the *Journal* and carrying on the work of the Society we must cut the *Journal* down to correspond. Under the new conditions we hope and expect to be able to improve the *Journal*.

Mr. W. Taylor: It just occurred to me whether it would be better to have a smaller number of members at the proposed increased subscription than a larger number at one guinea.

The Chairman: This resolution is, of course, a recommendation of the Finance Committee, which considers that it is sound financially.

The resolution was then put and carried, *nem con.*

The Chairman: I wish to draw the attention of members to resolution 29a, as follows:—

That the Council shall have power to elect "Corresponding Members of Council" resident abroad, provided that the member of Council so elected be a full member of the Society and has paid his subscription up to and for the year during which he is elected as a Member of Council; to whom matters of local importance, that is for the district for which he is elected, may be referred for consideration.

This is the form of the recommendation as originally drafted by the Council. To this, however, an amendment has been proposed by Messrs. W. A. Caldecott and S. H. Pearce, as follows:—

"29a. That the Council shall have power to elect each year 'Corresponding Members of the Council' from among full Members of the Society resident abroad. Each such official shall hold office during the year for which he is elected, provided, however, that his tenure of office shall cease as soon as he shall cease to reside and be in the country which he represents. The Council may refer to such officials matters concerning the country they respectively represent for consideration and advice."

The Secretary: The President, who is unfortunately absent, wished me to express his regret at being away, and his agreement with the amended proposal submitted.

Mr. W. A. Caldecott: This amendment was drafted by a sub-committee, consisting of Mr. S. H. Pearce, Mr. Ralph Stokes and myself, with the courteous assistance of our hon. legal advisers as to wording, and has been unanimously adopted by the Council in place of the original formal motion. By the amendment it will be possible for the Council to appoint at once a successor, in case of any "Corresponding member of Council" leaving the country he represents, so that the Society's work may be carried on without any break. It is to be hoped, that official representation of the Society in the chief gold and silver mining districts of the world will result in an increase of membership and likewise of papers and contributions to discussion from a standpoint, other than that with which we are familiar here. The appreciative references made to the Society's *Journal* and work, by those metallurgists whom I had the pleasure of meeting in Mexico and the Western States of America, were naturally a source of great pleasure to me. I trust that the endorsement by the Council and the Society of the suggestion made some months ago, to appoint representatives abroad, will result in still further extending the reciprocal benefits, which are available to all who are associated with our Society in its world wide scope.

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* The word "member" is here intended to mean also Associates and Students.

Mr. Ralph Stokes: I do not think it is necessary to add anything to what Mr. Caldecott has said. The proposal commends itself at once to anyone who knows what an interest is taken in this Society in all parts of the world, especially in the countries where there are big goldfields. I know that in Australia and New Zealand the *Journal* is very carefully read and followed, and I may say if we could get this Society represented there, they would take even more interest in our proceedings.

Prof. J. A. Wilkinson, on behalf of the Council, proposed that the original recommendation be withdrawn in favour of the amendment.

Mr. T. Lane Carter seconded, and the amendment as the substantive motion was carried unanimously.

It was also unanimously decided that the Council should be empowered to take the necessary steps to amend the Constitution and Rules accordingly.

The meeting then closed.

The Annual Dinner.

The annual dinner of the Society was held at the Carlton Hotel, on Saturday night, April 11th, the President, Professor Yates, presiding.

After the loyal toasts had been duly honoured, letters of apology for absence were read from His Excellency the High Commissioner, the Prime Minister, the Colonial Secretary, Sir George Farrar, M.L.A., Sir Percy Fitzpatrick, M.L.A., Mr. F. D. P. Chaplin, M.L.A., Mr. J. G. Hamilton, M.L.A., and others. There were also present:—

Messrs. A. Aiken (Hon. Auditor), F. F. Alexander, G. S. Burt Andrews (President of S.A.A.E.), W. H. Angus, R. W. R. Atkin, R. C. Atkinson, S. G. Bartlett, Dr. B. Bay, S. Beaton, C. A. Bevington, R. G. Bevington, F. Blaikie, H. C. L. Bloxham, S. H. Boright, W. Bradford, A. Brakhan, H. G. Brickhill, H. Briggs, M.L.A., J. W. Buckley, D. Buckeridge, T. P. E. Butt, W. A. Caldecott (Past-President), W. McC. Cameron, T. Lane Carter (Vice-President), P. Cazalet, J. S. Cellier, F. W. Cindel, R. Clarkson, A. C. Cochrane, W. F. Cosser, G. L. Craik (Secretary, Chamber of Mines), E. H. Croghan, W. Cullen (Past-President), J. S. Curtis, R. Curtis, Hon. W. Dalrymple, M.L.C., G. A. Darling, W. H. Dawe, A. Dewar, C. L. Dewar, J. H. Dinwoodie, W. R. Dowling, A. L. Edwards, P. L. Edwards, W. M. Epton (President, Transvaal Institute of Mechanical Engineers), S. Evans, J. F. Ferguson, J. S. Fisher, J. A. Frerichs, W. H. T. Frost, J. Gau, D. Gilmour, K. L. Graham, J. Gray, J. C. Greery, Jr., Thos. Greig (Secretary, Transvaal Institute of Mechanical Engineers), W. A. A. Hahn, H. Hamel, H. L. Harland, J. Hawthorne, H. Hay, H. Hellman, R. Henderson, A. J. Herald, S. H. Herbert, A. Heymann, J. Higham, C. H. Hilditch, C. B. Hilliard, W. G. Holford, E. Hollis, E. Homersham, R. Hosken, W. Hosken, M.L.A., A. G. Hoyer, — Hurlbatt, J. Hussey, C. E Hutton, J. l'Ons, Dr. L. G. Irvine, E. H. Johnson (Past-President), J. N. de Jongh, W. E. Kimber, C. B. Kingston, A. Kinkead, M. Knight, R. N. Kotze (Government Mining Engineer), Dr. F. E. T. Krause, M.L.A., W. F. Lance, M.L A. (Chairman, Transvaal University College), G. A. Lawson, H. Lea, J. Lea, Q. J. Leitch, C. D. Leslie, A. Lichtenstein, J. Littlejohn (Hon. Treasurer), Dr. Loeser, Dr. C. Lyons, Dr. J. McCrae, Q. C. McMillan, A. McNaughtan, J. N. Meeser, W. W. Mein (President, Association of Mine Managers), J. T. Milligan, A. Mitchell, J. T. Mitchell, W. E. C. Mitchell, D.S.O., Dr. J. Moir (Vice-President), P. T. Morrisby, Rev. J. O. Nash, S. Newton, R. Niven, H. Owen, S. H. Pearce (Past-President), Lionel Phillips (President, Chamber of Mines), R. G. Campbell Pitt (President, Rand Pioneers), Dr. C. Porter (M.O.H., Johannesburg), C. J. Price, T. R. Price, C.M.G. (General Manager, C.S.A.R.), J. F. Pyles, *Rand Daily Mail* Reporter, A. Rennie (President, Pharmaceutical Society), A. Richardson, G. C. Richardson, J. Robinson (Registrar, Transvaal University College), H. Rosendorff, Fred. Rowland (Secretary), C. B. Saner, Dr. J. Schlesinger, E. R. Schoch, A. D. Scott, G. O. Smart, Gordon Smith, B. C. T. Solly, B. Southwell, Prof. G. H. Stanley, *Star* Reporter, S. H. Steels, A. F. Stewart, R. Stokes, W. H. Stout, H. F. Strange, C. R. Stringer, H. Taylor, J. Telford, A. Thomas, C. K. Thomas, J. F. Thomas, W. S. Thomas, J. Thompson (Mayor of Johannesburg), S. C. Thomson, H. T. E. Thorpe, W. E. Thorpe, *Transvaal Leader* Reporter, W. K. Tucker, M.L.A., J. A. Vaughan, Hon. J. de Villiers, M.L.A. (Minister of Mines), W. H. Visser, J. H. Vivian, J. F. Walker, R. C. Warriner, T. Watson, G. E. Webber, E. M. Weston, A. Whitby, H. A. White, Prof. J. A. Wilkinson, E. Williams, J. R. Williams (Past-President) and L. J. Wilmoth.

Dr. F. E. T. Krause, M.L.A., in proposing the toast of "The Chemical, Metallurgical and Mining Society of South Africa," said:—

"It is not the first time that I have had the pleasure of gathering round the festive board with you. It was quite in the infancy of your Society that I first came to Johannesburg, and I was the invited guest of one of your first Presidents. To-night I assure you I appreciate most highly the compliment you have paid me, and not only myself but I think also the Mining Regulations Commission, of which I have the honour to be chairman, in asking me to propose this most important toast, yourselves, the Chemical, Metallurgical and Mining Society of South Africa. Your Society, as far as I can gather, is one of the most important and largest in South Africa. Your objects are to a very great extent scientific and technical, and you are in the happy position that you have absolutely nothing to do with politics. The latter are, I understand, absolutely tabooed, and I congratulate you on that happy position, and I only wish that I myself belonged to such a Society. But on the occasion of an annual dinner politics seem to creep in merely because they seem to be everywhere, and if political as well as economic questions, are not discussed by you, are at any rate listened to by you in the form of addresses on those particular subjects. Anyhow, to-night I am not going to discuss politics, nor am I going to discuss economic questions. I think there are other speakers from whom we expect to hear more about those subjects than from me.

In going through the addresses last year I was struck with the pessimistic forebodings in the speech of the gentleman who proposed the toast of "The Prosperity of South Africa," and what specially struck me was, that the gentleman who spoke seemed to think that we were not yet at the end of the long lane of depression. He spoke also of the native labour supply, and I have taken down a few of his words. He said, 'There was at the present time a momentary surplus, but experience teaches us that it is only a passing phase, and that this condition of affairs will not continue for long.' I am glad to say that in reading the other day a speech of Mr. Perry's at the annual meeting of the W.N.L.A., that these gloomy forebodings have not been realised. Mr. Perry thought that they would not be realised, and that we are in the happy position that our native labour supply is more than sufficient for the requirements. I only mention this, because I understand that most of the members of your Society are directly or indirectly connected with mining, and, of course, you cannot carry on your work, unless you have the unskilled labour available. In going through the Proceedings of your Society, I was struck by the cordial relations which exist between the members. One finds, that while the criticisms passed on the papers are severe—in fact, though there is some very hard hitting in some of those criticisms—there is an absence of personalities, and the fair and reasonable nature of the criticism is obvious. It indicates, that those who take part in the discussions are not concerned with the personality of the speaker but merely with his opinions. It has been through the efforts of this Society, that the metallurgy of gold mining has reached such a high state of perfection in this country. It is not very long ago that the cyanide process was one which, it was thought could not be developed to the extent that it has been. And I take it that it has been through the efforts of your Society that we can say to-day that the cyanide process has reached its highest development in South Africa, a development which it has not attained in any other country in the world. What was formerly considered a good residue fit for the dump is now an original sand worth working in a profitable manner. But had it not been for a kindred society — the Engineers' Society — I do not think you would have been able to have worked so efficiently. When you devised improvements and new inventions it was through the engineers, who devised the necessary machinery to carry out your ideas that everything was brought to such a successful issue, and I think some recognition is due to them for the assistance they have given you.

I think also that the heads of the mining industry have shown that they recognise the good work that you have done. I am glad to say, that the impression I had, that the heads of the mining industry were only concerned with extracting the last grain of gold out of the ore was an erroneous one, and that they also are capable of recognising the good work done by others. We need not go into the details of the improvements your members have introduced, which have not been concerned alone with the useful side of mining, but you have considered what was of a wider and broader nature, your duty not only to the mine owner but also to the mine worker. I hardly think it is necessary for me to remind you of the papers, which have been read on safety measures in mining—miners' phthisis, noxious gases, and the compulsory introduction of cyanide antidote. All these things indicate and clearly show that your members are also humane people, and look on the humane side of mining. Your Society has gone further than that, its scope of activity is still wider, because I also find that you busy yourselves with the educational side of mining. I find, for example, that papers have been read on the education of miners. I think your President is very much concerned with that particular subject, judging from the evidence he has given before my commission, and your Society by busying itself with that subject is doing a great and noble work for this country. Members of your Society are also doing public work. I need hardly remind you that members of your Society have done good work on many of the commissions appointed by the Government, notably the Miners' Phthisis Commission, the Safety Catch Commission, the Single Outlet Commission, and at the present moment the Mines Regulation Commission. I am wedged in here by two of them, and therefore I think it shows that not only from a private but also from a public point of view you do work which is recognised not only by the mine owners but by the Government of this country. The educational work that members of your Society are doing is, I assure you, of a very useful nature. During the last few months I have been sitting on the commission I have learnt more about CO_2 than I ever knew before, and when I came into the room to night the first thing one of your members did was to try and pump CO_2 into me. I do not think that the usefulness of your Society is to be entirely confined to gold mining. There is a much larger industry in my opinion, which is going to be developed in this country. The more we look for tin and other base metals the more we seem to find them. I quite admit that some of these metals are only found in the imagination. Some are only found in prospectuses, others again you will only find in shares, and others again you will only find in the Stock Exchange. At any rate, you come to the conclusion from these indications that tin, for instance, must be somewhere. And you find other base metals about if you only look for them in a proper manner. You are quite aware of the fact that the processes of reducing these ores are in a very imperfect state, and much is required of you in order to assist us to properly develop this industry. There is no reason, as far as I can see, why we should send our base-metal concentrates to Europe. If you consider the larger requirements of the mining industry as far as tin and lead are concerned, I think it will be patent to every one that it is worth while for the capitalists to go in for that particular class of investment. And at once I may say that, as far as the educational factor of your Society goes, it is time to teach our young people that branch of the industry, too. I see the Chairman of the Transvaal University College Council is present, and I think it is time they got the necessary equipment for this branch of work. Once you have taken on responsibilities they lead to others, and that is the case with your Society. Do not rest satisfied with the work that you have done, do not be satisfied with your past laurels, but look to the future, and in the future gain other laurels, laurels that I am certain you will gain, and your Society will in the future be a greater success than in the past.

The President, replying to the toast, said: As your President it falls to me to thank Dr. Krause for his eulogistic toast to our Society. We have made steady progress since our foundation in 1894, and our membership of over 1,100 includes, I am pleased to say, most of the departmental heads along the reef, as well as the majority of our managers, consulting engineers and financiers. We have members in most of the world's mining camps, and as an illustration of the far reaching nature of our work I would point out that during the past twelve months we have received papers and other contributions from the South Pacific, Great Britain, Western Australia, Austria, the United States, the Malay Peninsula, and the Arabian desert. It is a very great pleasure to the Council to frequently receive from many lands tributes to value of our discussions, discussions which continue to be noted for their freedom and thoroughness, and in this connection I may say that I am inclined to think that, after all, the warmth which is introduced into our meetings by our members taking off their coats, metaphorically speaking, is advantageous. To my predecessors, and to the past and present members of the Council, and last, but not least, to our Secretary, Mr. Rowland, is due the credit for making the Society what it is to-day. It gave me especial pleasure to hear Dr. Krause's remark that we, as a body, give considerable attention to the welfare of the workers along the reef. During one of our recent Saturday excursions Mr. Dalrymple, whose guests we were, remarked that he did not think scientific societies were sufficiently appreciated by the community. I agree with him. I do not think the value of their work has been properly gauged. Take this Society for example. Dr. Krause has mentioned that that wonderful process, the cyanide process, was largely developed by our members. Our papers and discussions have largely taught our millmen and cyaniders, and now they are teaching our underground workers. Our business is to elicit information and distribute it broadcast for the good of all concerned. thus educating them and increasing their efficiency, and this we do by means of our papers and discussions which keep them in touch with the latest ideas and practice, and thus enable them to keep abreast of the times. The technical societies of Johannesburg supplement the work of the Transvaal University College; they pick up the tow rope of education when the College throws it overboard, and they help workers to make headway, and avoid shipwreck on the rocks of ignorance and inefficiency. We must have efficiency on these fields, and efficiency is synonymous with education. Of what use is it spending £10,000 to discover the best rock drill if you cannot provide competent men to operate it? Already we are finding what an important factor is the man at the handle of the machine. I submit that those prominent countries, the United States, the United Kingdom and Germany, owe their position in the world largely to the high standard of education and efficiency of their people. I submit that education makes better men, better citizens, and cheaper workers, and it is largely to the credit of the Transvaal Government that it is displaying such an interest in this direction. Our farmers must be taught how to do justice to the land, and our miners must be taught how to break ground The latent resources of this country are truly enormous, but a skilful people is needed, if we are to benefit to the full. I am afraid that I have drifted into preaching the gospel of education and efficiency, but I submit as my excuse that education has been a power in the past, that it will be a greater power in the future, and that the influence of this and other Societies, as educational factors, will increase as the years progress.

Hon. J. De Villiers, M.L.A. (Minister of Mines), in proposing the toast of South Africa, said: When I saw the distinguished company that is here present to-night, men who are eminent in science, and who have made the Witwatersrand what it is, the greatest gold producer in the world, I feel that it is a great honour and privilege indeed for me to have been asked to propose the toast that has been entrusted to me. It is on occasions such as these that one wishes, not indeed that one had the power of speech possessed by Demosthenes, as was wished by my friend, Mr. Neser, the other night, but that one had the command of language and the power of expression, which a brother scientist recently showed when he lucidly explained that the determination of sex was due to the anabolism and catabolism of the protoplasm. I shall have to content myself with less lofty and less formidable language. In proposing this toast the orthodox way of handling it is for the speaker to trace the causes of the depression through which South Africa is passing, and then to attempt to find a remedy according to the taste and fancy of the speaker. As a rule, he paints the depression in more or less lurid colours, generally I find that it is rather more than less lurid, and South Africa is usually painted as being at death's door. Now I wish most emphatically to protest against that view. To my mind, South Africa is sound to the core, and as a matter of fact, I think it may be said to be at the present time convalescent. A few years ago we had an attack of great extravagance and folly. We mistook that attack for signs of exuberant health, whereas, as a matter of fact, that was the real disease. Now we have come back to our sober senses, and South Africa, as I said before, is convalescent. I am not going to suggest any remedies. I find in my experience that the man who has a sure cure for anything is generally a quack. I am not going to attempt that impossible task to-night, but I do wish to discuss one or two points with you, which are of especial importance to you as residents of Johannesburg. The first point is the question of the unemployed. I wish to say here that it has been a great pleasure to the Government to see the hearty co-operation that was evinced by the public of Johannesburg through the Mayor, and by the Chamber of Mines through its President in assisting the Government to solve this very knotty problem. It is one of those questions, which one might almost call insoluble. I do not expect that we shall be able to solve the unemployed problem entirely, but what I do expect, and what I think I have fair grounds for believing is, that we shall be able to grapple with this problem and come to close quarters with it. So that finally we can sift the "Weary Willies" who will not work from the men who will work. A white man who wants to work is a great asset to this country, and we want to do everything in our power to keep that asset here. But we must also make those people, who will not work, understand, that they will meet with no sympathy from any of us. There are two directions in which the Government is working at the present time and where we expect the hearty co operation of everyone who is here to-night. You have all heard that the Government intends to start a mining school, to acquire a mine and to train boys of this country, no matter what their origin, whether they are British, Dutch, French or Germans, to become skilled miners. The Government feels that the mining industry has been in this country for 20 years, but the boys of this country have not had that share in the development of the mining industry that they are fairly entitled to claim, and the Government wants with your co-operation to give the young men of this country, who want to go in for mining, the opportunity of becoming skilled miners in time. We propose to acquire a mine and to make young men of 16 years and upwards go through a thorough course of training of three years, so that at the end of that time they can be turned out as skilled miners. But there is another direction in which the Government would like your co-operation. You will see that this is a very slow process, and the Government is very anxious that you should catch the young men who are now growing up and prevent them from also being on the streets in time, that you should give them the opportunity of going on your mines at present, and if they do not become skilled, they may become semi-skilled and be able to do some work on the mines. If the mine managers would assist the Government and take the youths who are growing up on to the mines they will be doing this country a permanent good, because they will absorb all the young men who want to come in and prevent them afterwards from walking about the streets in the shape of unemployed in the future. I do not propose to touch upon the question of white labour, but I wish to say this, that the old idea that used to be prevalent on the Rand, that work is derogatory to a white man, is past; it has been exploded. There is no kind of work that is derogatory to the dignity of anyone. If a man takes off his coat and works, he is bound to become a better man than if he loafs about the streets. And if we realise in time, that there ought to come a class of workers here who are somewhat skilled, and even unskilled, if we realise that and help it on, we shall be doing a great good to this country. I have been told, that amongst a certain portion of the workers of the Rand, there is hostility to this scheme of the Government. For the life of me I cannot see what the objection of the skilled miner can be to this. The Government does not propose, that he should be ousted from his position. He ought to welcome any competition, and he ought to welcome the fact, that the Government wants to train other skilled miners, because that certainly will not diminish his pay. And, moreover, if it is understood that the mines will also employ other men as somewhat skilled workers and still others who are unskilled at a less wage I do not see how it can possibly affect any skilled miner on the Rand. There will be an incentive, of course, for every man to work himself up, and in the course of time, if it is a man who will do his best, he will become a skilled miner, and he will naturally command the highest wage. But I want to say this to-night, that, whether there is opposition to the scheme or not, the Government means to go through with it. The other point is a question of the very greatest importance to you as South Africans. I address you advisedly as South Africans, because that is the point of view that is growing every day, and that is the point of view that I would like to impress upon you. Your President to-night has used a word which came home to me. He spoke of Johannesburg and other places, where mining is carried on, as mining camps. Johannesburg has long ago grown out of that. It is now the home of a permanent population. I shall leave this idea to be developed by Mr. Lionel Phillips, who will respond to the toast, but I want to say one or two words about it. You will have noticed that there are gradually forming two schools in South Africa upon this question of Unification or Federation. The one is for Federation and the other is for Unification. It always seems to me advisable in matters of this kind to take the larger view, to go for the highest, and if you find eventually that that is impossible of attainment, then do the next best thing, but at any rate strive to attain the highest. What I want to warn you against is this. Do not let us in the heat of debate gradually make these two schools oppose one another, because that will do the cause of closer union, which we all have at heart, more harm than good. If we cannot see eye to eye, if some are for Federation and some are for Unification, do not let us quarrel with one another about it, but let us try to understand each other's point of view. To my mind, we should go for the larger view, and strive for Unification. I think the question is not impossible of solution, if we go by gradual stages. Even if we federate on customs and railways, as I hope we are going to do, or with a court of appeal, then we shall be working in the direction of that high ideal for which we are all striving. I only want to say a few more words. I have been, as you know, for the last few days in the Congress of Het Volk in Pretoria and what has struck me more than anything else there, was the feeling which was evinced amongst the people, who come, most of them, from the country, towards the population in Johannesburg. That is, to my mind, one of the most hopeful signs of the times. There is a new spirit abroad in this country. There is a spirit of co-operation and of toleration, and it is for us, gentlemen, to take advantage of that spirit. Do not let us do anything now that will retard the progress of South Africa.

Mr. Lionel Phillips (President, Transvaal Chamber of Mines), replying to the toast, said: I do not propose to follow the Minister of Mines in his scientific excursion, but I will come down, more particularly, as I am responding for South Africa, to some of the other matters to which the Minister has referred. He has spoken to you of the unemployed question, and I do not propose to follow him at length on that subject, because to me this is hardly the occasion, and, in addition to that, it will be my duty in the course of a few days and in another place to make reference to that subject. But this I may say to-night, that the mines will do their best. We want the men who are taken on to do their best. We want the men who are in the mines to-day to give the other men a chance to do their best. And if we all work in that spirit, I think much can be done to alleviate the distress which, unfortunately, for the moment exists in this place. I believe it is only transitory, and I believe we merely have to study it, to deal with it with discretion, and we shall find, that it will pass. I am going to turn at once, with your permission, to that which is much deeper in my heart, and which, I believe, will be much deeper in your minds, a subject that the Minister of Mines merely referred to—the subject of closer union. Speaking for South Africa, it is at the moment the question of all others that occupies the minds of those who have the welfare of this great sub-continent at heart—those who think. I believe that there is a consensus of opinion in favour of Unification over Federation. It appeals to the imagination, and it is a higher ideal. Everyone, who speaks, hopes that we might have Unification if the difficulties are not too great, if it can be done—as they generally say. Well, I admit that the difficulties, either of Unification or of Federation, are supreme. They are so difficult, indeed, that I believe, on the whole, the easier thing to accomplish is the bigger one of Unification. I have studied, as many of you no doubt have done, a series of articles that was written recently by Mr. Hofmeyr, the Clerk to the Legislative Assembly. I think, in writing those articles, Mr. Hofmeyr has rendered a public service, for he has treated the matter in a thoughtful and dispassionate spirit. Mr. Hofmeyr exposes the difficulties in the path, difficulties of a stupendous nature, but I think that very exposition of the difficulties is an argument in favour of Unification. Mr. Hofmeyr suggests that we should have the stepping stones of Federation. Now, for my single self, I think that stepping stones in questions of this kind are exceedingly dangerous instruments to use. If we put a stepping stone in mid-stream we shall find every Colony in South Africa having other stepping stones to put beside it. Some of these stepping stones must be removed and replaced by others, and the result may be, that we will get a pile of stepping stones in the middle of the stream that might be washed away by the first flood that comes down. It is one of those cases, I think, in which we require a little boldness and spirit. Of course, we must examine with care the width of the stream that has to be crossed,—we must examine the respective banks. But when we have done that, I say it is a case of taking our courage in both hands and leaping over, as the intrepid horseman does in the hunting field. I propose to drop metaphor, because this is a subject which should be dealt with in very plain language. Any act of confederation, that we can conceive, must leave to the Colonies, as they exist to-day, a certain measure of power, and that means that it must give to the supreme Parliament only a certain degree of power. For example, if one Colony should say it desired to reserve in its own hands the native question, another the subject of education, another the manner in which the land laws were administered, and another still the control of the mining industry, then natives and education and land and mining would have to be excluded from the control of the supreme Parliament. I leave out for a moment, and advisedly, those questions of customs and railways which, we can take it as a *sine quâ non*, every Colony, as it at present exists, would like to have the control of in its own hands. If we can imagine a federation with certain things excluded, who so venturesome as to say that the various Colonies, who are parties to the Federation, will be prepared as time goes on to divest themselves of further of their powers to be entrusted to that central authority? I believe it is the experience of the world, that they would rather endeavour to take away from that central authority something that had been given, than invest it with still further powers. The division, for instance, of customs and of railway revenue, the basis upon which taxation is to be raised, may in the course of time, if we have Federation, prove that one particular Colony is gaining an advantage, and if that Colony did gain an advantage, the other Colonies would find that they suffered a disadvantage. This alone would be a fruitful source of friction. If the revenue had to be raised separately, each of their Colonies would desire to expend that revenue separately. It seems to me, therefore, when we face the problem, as we must do, as a whole, we shall find, that we either have to raise our taxation separately and spend it separately, or raise it unitedly and spend it unitedly. And this is the fundamental difference between Federation and Unification. I do not propose to address you on the very difficult and important native question, but let us take it as an example, and ask ourselves, if we hope to solve it as a South African rather than as a colonial question, why we should not solve it now? Are we likely later on to find the views of the inhabitants of this country less divergent than we were to-day? It seems to me that it is a supremely difficult question, but we might just as well face it now as at some future time, and it is quite conceivable that a central Parliament, established with Unification, would find it desirable to frame regulations applicable to different portions of South Africa, as far as the natives are concerned, in accordance with the degree of civilisation that they, in those particular parts, may have attained. One of the very serious difficulties which was raised in a personal conversation with me by a well-known statesman in the Cape Colony was this; he said: "How do you propose to bridge over the interim between the abolition of the existing State Parliaments and the time when the central Parliament will be able so to decentralise that it can take the government of the distant parts of South Africa in hand?" I admitted that it was a difficult question, but I did not believe an insoluble one. Let us for a moment imagine that, previous to the abolition of the existing Parliaments, a council should be formed, consisting of, say, two members on the Government side, two members of the Opposition, and a judge of the Supreme Court, preferably the Chief Justice, entrusted with authority to carry on the government of the country under the laws as they exist. It seems to me that that might be a workable system. I offer it merely in the humblest of spirits, but I offer it as a possible means of bridging over the interim. We must remember that this council would not have to undertake any legislative work. It would be carrying on the administration of the country. In fact, it would merely be a representative institution to see that the heads of departments acted in accordance with the law as it stood at the time. And assuming that this Council found itself faced with some knotty or difficult point, it could always refer in that case for the decision of this central executive. I, therefore, think that although the difficulty is great, it is not beyond the scope of human ingenuity to frame some means wherby the interval could be satisfactorily provided for. Of course, men of Cabinet rank would probably be found—as I am sure they would be—in the central Parliament, so that they would not be available as members of that council. But assuming it is difficult to bridge the interval, I say that, however complex it may be, it pales into insignificance beside the infinitely greater difficulty of apportioning revenue and expenditure between the various Colonies in South Africa in a federation. We have many warning lessons before us. In the United States of America we see litigation and constant friction between State Parliaments and the Senate. We have in Australia the heartburning and ill-feeling, that exist between the Federal authority and the State Governments. And I say, let us in South Africa take warning by these examples and let us endeavour to avoid similar conditions The proposition to put to members is this: Do we desire to coalesce? have we got the ambition to see the united Colonies of South Africa established? are we imbued with the idea of the big nation versus the small communities? It is a much more inspiring object, and anyone whose mind is not pervaded by the parochial instinct must have little hesitation in making his selection. There are people who say, "You will leap in the dark into unification and give a blank cheque." Well, personally I think that is a false description. It is quite true, that in the first years of unification it might be desirable to protect the hinterland in some measure.

It is conceivable, when we hold in view the disparity between the population, for instance of the Cape Colony and the population, say, of the Orange River Colony and the Transvaal combined, that a combination of the representatives of the population in the Cape Colony might do something in the central Parliament that would be detrimental to the interests of the hinterland, and that being the case, it is probably desirable that some special representation should be accorded to the hinterland Colonies in the first instance. But though this may be a desirable thing, it does not mean that we should not at once fix the basis upon which every one is to have rights in the future, which should automatically come into being in the course of a given number of years. We must remember this: once we have expunged these Colonial boundaries, then Colonial patriotism and Colonial interests will cease to exist. We shall have the one paramount South African interest, which will pervade the minds of the whole of the people of the sub-continent. And, of course, as the potentialities of this vast continent are developed we shall find population gradually shifting, or, I hope, a new population coming to those centres where commodities can best be produced. That in itself, of course, would alter the centre of power in South Africa as a whole. But about unification one thing is certain: that it will produce a stability we do not enjoy to-day, it will tend to promote economy in government, it will be the means of our having a general broad policy of government, it will improve the credit of South Africa as a whole, and it will inspire people with confidence to come into this country and invest their money.

There are those gentlemen—we read of them daily in the newspapers—who desire to see built round the Transvaal a ring fence. These gentlemen desire to have protection. They desire to see certain industries established here before there is any system of union. Well, it seems to me that the reply is very simple. If we put customs houses round the Transvaal and do damage to our neighbours—who are three times as numerous as ourselves—our neighbours might suffer it for a time. But as surely as we are present in this room, if we damage our neighbours enough, we will either have to pull these customs houses down at their instigation or our neighbours will pull them down for us. We are a part of South Africa, wedded beyond recall, and divorce is out of the question. The question is, Are you going to be happy partners or are you going to be fractious neighbours?

I wish you for one moment to imagine South Africa vivified. Speaking of the past conduct of civilised man since he passed the frowning heights of Table Mountain, and I wish I had the voice to speak as it would be spoken, if these words came through the mouth of a swollen Orange River, I imagine the torrent of unanswerable reproach that would be heard. What has man done in the arts of peace? I am afraid the record is a tale of ignorance and lazy indifference. I am not speaking, mind, of the causes but of the effects. We all know that there is every reason why the first settlers who came into South Africa finally found themselves upon the best pieces of alluvial soil producing the bare necessaries of life in the easiest possible manner. We know it was due to the pests, the wild beasts, the savages, the various difficulties and the absence of markets, and that no matter what people had been brought here, or from what part of the world, the results would have been precisely the same as they were under the conditions of those men brought here in the first instance by the East India Company. I am not, therefore, saying anything against the early settlers in this country, but I am speaking of the facts as they are. In the arts of peace but little progress was made. Of course, to-day we have the assistance of highly trained intelligence in agriculture, and so much demonstration has been made already that it is safe to say we shall see an enormous development in the surface of the land—a development perhaps in which energy and industry will be as great as, or greater than, that which has been bestowed on the mining industry in seeking out the wealth of nature buried beneath the surface. And that is something in regard to which my scientific hearers may also assist, because the chemistry of the soil is one of those things that has to be studied minutely. When, therefore, they have a little spare time from those arduous duties which I am afraid I am connected with imposing on them, perhaps they will also direct their attention to some of those problems which may help to make the surface of this country more productive than it is to-day. I want to turn for a moment to another side, to leave the arts of peace, and ask you to think of the record we have from the earliest days in another direction. And what do we find? We find practically from the days of Van Riebeek men's thoughts were turned against each other, that they from time to time burst into open conflict, the result of which was that the fair surface of the land was drenched with human blood. It was a record of which everyone need be heartily ashamed. The blame rests not with one Government in this country or with one Government across the sea, nor does it rest upon one, but upon both the principal representative European races that settled in this country. I do not rake up the past, because but little useful purpose can be served by that, except with one object in view, and that is that we may learn the lesson and act differently in the future.

The Minister of Mines has spoken of the better spirit, and I am sure he is right. There is a spirit of fraternity about that certainly did not exist in South Africa in years gone by. Perhaps we may wipe out the past in time. Anyhow, we can do our best to atone for it, and there is one direction more than any other in which we can do that, and it is by forcing on, by using our very best energies and powers to bring about the union of all the Colonies of South Africa. Let us begin by blotting out the boundaries and by destroying those divisions which were senselessly and wrongfully built up in the past. They have caused most of the past suffering, and if they are left in being they will very likely repeat that sinister rôle and produce more suffering in the future. Union is a delicate plant, and its seed must be sown in faith. We are not going to achieve union by an ignoble squabble upon the relative interests of persons in the country, nor shall we get it by floundering in a quagmire of details. We shall have to approach the question upon a high plane and in the spirit of fraternal amity. The foundations must be laid in principles that are sound. There must be a guarantee to all civilised men in this country of equal rights, and preferential treatment of any one must be avoided as the plague. The day we set the seal upon the union of South Africa will be the birthday of a great nation. It can be done and must be done, and it will be done if men's minds are not accursed with a puny spirit and are tuned to the requisite heights. There are a few splendid lines I wish to conclude with, and I hope you will ponder over them, and that the words will sink deep into your hearts. They are these :—

"It takes a soul to move a body,
It takes a high-souled man to move the masses to a cleaner sty,
It takes the ideal to blow a hair's breadth off the dust of the actual."

Mr. T. Lane Carter next proposed the toast of "Our Guests" and "Kindred Societies." In doing so he said :—There is one thing that we have to be thankful for in regard to our kindred societies, and that is the friendly spirit that pervades the members. We all have one object in view, and that is the advancement of this great industry. Every member of every society always has done and always will do his level best to advance that cause. I have also to propose the health of our guests. We are very glad to have with us to-night Mr. Lance, who, as you know, is Chairman of the Transvaal University College. Kindred societies are bound to take a very great interest in the work that the College stands for, as the future success of kindred societies rests to a very great extent on the kind of men that they turn out at that University. May I say that it is the hope of kindred societies that there may grow up in this Colony not small schools in several different centres but a great University, which will be able to hold up its head amongst the best in the world. We also have with us Mr. Burt Andrews, our Town Engineer, who, as you know, is also President of the South African Association of Engineers. I have been told, that he has been affected with a microbe, which seems to be permeating the Transvaal very thickly, and that is the *bacillus amalgamationis.* I do not wish to say that he has any idea of amalgamating the city of Johannesburg with any of the surrounding mines or anything of that sort, but I understand that he is very anxious to amalgamate his association with a kindred society. That has been discussed for a long time. Many of us have sympathised a great deal with Mr. Andrews in the many serious and difficult problems that he has had to tackle, and one of them has been unskilled labour. I think we can call Mr. Andrews the geologists' friend, for during the last few months the most beautiful prospecting trenches have been cut in the streets, and I firmly believe that the geologists instead of walking up the streets have gone up through those trenches and investigated the strata which underlies this great metropolis.

Mr. W. F. Lance, M.L.A. :—On behalf of the guests present I have to thank you for the genial way in which this toast has been proposed and received to-night. I wish also to express our high appreciation of the honour conferred upon us in the invitation to be present at this very interesting function, and also for the generous hospitality we have received at the hands of our hosts, one of the oldest and most influential of the scientific societies of the Witwatersrand. It has been often said that we are nothing if not scientific, and where indeed should science find a home if not in Johannesburg, where the overwhelming importance of applied science as a factor in the development of our mining industry comes home to every one of us. Reference has been made to-night to the educational factor in the work of this Society, and I claim a very close connection between this Society and the college which I have the honour to represent, not only on account of your own most able work in both directions, but because the one stands for the best or some of the best of scientific thought and work, while the other stands for that scientific and technical work which necessarily leads up to and waits upon the higher work. And I hope that this close connection will shortly be emphasised in a more material form in the new buildings now in course of erection on Plein Square, where we hope to provide, at all events, a certain amount of accommodation for the different scientific and technical societies together with a suitable lecture hall for their requirements. And whatever may be the future of higher education amongst us—and these matters are still under consideration, and it is difficult to say what may be ultimately evolved—but whatever that may be, and whatever may become of what we regard as more strictly University work, yet the trusts upon which the Plein Square site is held will always, if properly carried out, ensure that that will remain permanently a centre for science and scientific thought, and a home for technical education of every grade and a centre which will radiate the best of these things for the benefit of this country. I have referred to the close connection of applied science with our mining industry, but our thoughts naturally turn, as has indeed been suggested already this evening, in other directions. We talk of closer settlement of the land and we recognise what has already been done in the way of the application of science to the pursuits of agriculture and stock raising. We know the excellent work that has been done by our experimental farms and our bacteriological institute, but we recognise at the same time with regret that we have not as yet provided for scientific and technical education in that direction, and those of our youths who to-day would make the land their life occupation are driven, either to start upon that walk in life insufficiently equipped with scientific knowledge, or are forced to go to other places, within our own knowledge, as far as Canada and the United States to get that scientific education which should be better given on the spot. This question of an Agricultural College has been brought to the notice at different times not only of the present Government but also of its predecessor by the Director of Agriculture and by the University College, and it is with regret that one finds that nothing has so far been done towards a practical solution of the matter. I feel that no excuse is needed for drawing attention to the matter to-night, and I do so in the hope that it will not be long before our Government takes the initiative in this important matter.

Mr. G. S. Burt Andrews, replying on behalf of kindred societies, said :—I feel that one of the most difficult tasks in connection with the presidency of the association is to follow on after a flood of eloquence, such as that we have listened to to-night with the greatest interest and pleasure. It is a great pleasure to kindred societies to know that your Society has made such excellent progress in such a comparatively short time. I think that kindred societies have to agree, whether they like it or not, that this is one of the strongest on the Rand at present, and we feel that the strength of this Society is likely to improve the position of the other institutions and societies on the Rand, because I think I can safely say that there is a rivalry and a healthy rivalry in connection with the different associations in Johannesburg. There can be no doubt as to the usefulness of these societies. A great deal of interest is taken by scientific men on the Rand in the different problems, which they have to deal with, and I think that everyone will realise that there is a greater scope for originality in Johannesburg than there is in older established countries. In Malthus' work on lost arts, he says that the best steel is the greatest triumph of metallurgy, and metallurgy is the glory of chemistry. I do not know whether any of you are prepared to contradict that, but I believe it is a solid fact. At the same time, although the metallurgy of gold is not perhaps so difficult or so interesting as the triumph brought about by the best steel, still I think we can say that the metallurgy of gold on the Witwatersrand is second to none in any part of the world. I think that is largely due, not only to your Society, but without wishing to pat kindred societies on the back, to the kindred societies themselves. It has been said more than once that very little interest is taken in the scientific societies of Johannesburg. I should like to contradict that in the very strongest manner, because I think, there are very few towns in the world where more interest is taken by the scientific societies than in Johannesburg. I think I am right in saying that more than one meeting a week takes place here of a scientific society, which means 52 papers or more on scientific subjects during the year. These papers are discussed in a whole-hearted manner and, as a previous speaker has said, without any personality. I think a good deal of the success, which has been achieved in Johannesburg, is due to the discussions which have taken place in the different societies. You have been hearing about Unification and Federation, and I should like to talk about amalgamation. For some years past it has occurred to me, that a great deal more good could be done in this town, if some of the big societies would amalgamate. There is a scheme on foot just now to amalgamate the South African Association of Engineers and the Transvaal Institute of Mechanical Engineers. One of the chief reasons for this is, that their interests are very much the same. Many of our scientific gentlemen on the Rand belong to all the societies, and when they want to read a paper it is not an uncommon thing for them to turn over in their minds to which society they will bring it. Now, that is altogether wrong. The societies should be brought into closer union. We could save administrative expenses and other things, and so bring the whole thing into one homogeneous institution. I think if the societies amalgamated we should have greater recognition on the part of the Government, and on the part of the public. I am quite sure that when anything has to be referred to any particular society, it must be very difficult for anybody to make up his mind as to which society to refer it to. If all the societies were amalgamated this difficulty would be overcome.

Hon. W. Dalrymple, M.L.C., proposed the health of the Chairman, of whom he spoke as one filled with vitality, capacity and enthusiasm, to which Prof. Yates briefly replied, and the proceedings ended.

Obituary.

It is with regret that we have to report the death of Mr. W. V. Hunt, an associate of this Society, which occurred at the Salisbury Hospital from enteric fever on the 19th February. The deceased gentleman, who was a native of Ireland, had been in Rhodesia for some time, and until recently was battery manager of the Surprise mine, Selukwe. A few months ago he went out to the Kimberley Mazoe mine as manager, and was brought into hospital about three weeks before he died. Mr. Hunt was 37 years of age and leaves a wife and four children. He was admitted an associate in October, 1904.

It is also with regret that we announce the death of Mr. John Grove Johnson, F.C.S., an associate of this Society, which occurred suddenly towards the end of March. Mr. Johnson, who was well known to many of the older members of the Society, was the senior director in the firm of Johnson & Sons' Smelting Works, Ltd. (formerly Johnson & Sons).

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

The Assay of Slimes Residues.*—"As the result of much experimental work KCy is now used as a reducing agent, in preference to any of the carbon compounds, for the assaying of residues resulting from the roasting and subsequent treatment of ore from the mine. This practice has been found to yield better results than the usual methods employed, and particularly is this the case with residues containing a large proportion of soluble gold. The gold in the slags from the usual method of assaying can be recovered by fusion with KCy and litharge, and the advantageous use of this reagent in assaying the various products of a gold mine, and in the smelting of zinc-box sludges, is being actively investigated.

The following assay results were obtained operating with a specially prepared residue containing much soluble gold. A filter press was charged with current pulp, and the resultant cakes of residues were dried with compressed air.

1. The smelting charge used was:—

Residues (dried at 100° C.)...	100 parts.
Soda ash...	50 ,,
Borax glass	12 ,,
Litharge...	150 ,,
Flour	8 ,,

2. The smelting charge used was:—

Residues (dried at 100° C.)...	100 parts.
Soda ash...	50 ,,
Borax glass	12 ,,
Litharge...	150 ,,
Sodium cyanide	17 ,,

3. The residues were triturated and washed on a suction filter, the smelting charge for the dried residues resulting from this treatment being the same as in 1; the gold in the solution being recovered by evaporation with some zinc shavings, followed by the usual scorification, etc.

Grains Gold per ton:

1.	2.	3.
		Insol. + Soluble.
27·9	34·49	6·31 + 28·2 = 34·51.

The above results are the averages of a series of determinations made by the different methods enumerated."—*Journal of Chamber of Mines*, Western Australia, Sept. 30, 1907, p. 841, and Oct. 31, 1907, p. 901. (W. A. C.)

Gaseous Nitrogen Trioxide. — "A special interest is at the present time attached to the chemistry of the oxides of nitrogen owing to the important part played by these substances in the commercial process for the utilisation of atmospheric nitrogen by its conversion into the nitrates of the alkalis and alkaline earths. There has always been a certain doubt as to the existence of nitrogen trioxide in the gaseous state, although this oxide is known at low temperatures in the liquid condition. In 1885 Sir William Ramsay and Mr. Cundall showed that in the main the gases derived from this liquid trioxide consisted of nitric oxide NO, and nitrogen peroxide partly in the complex form N_2O_4 and partly in the simpler form of NO_2. Later, in 1899, Prof. H. B Dixon and Mr. Peterkin succeeded in demonstrating that such a mixture does contain a small proportion of gaseous nitrogen trioxide, N_2O_3, and that accordingly the reaction $N_2O_3 \rightleftarrows NO + NO_2$ is a reversible one. This reversibility affords an easy explanation of the behaviour of the mixed gases towards various reagents. For example, these gases dissolve in cold concentrated sulphuric acid to form nitrosulphonic acid. In this reaction, which occurs in the Gay-Lussac tower of the sulphuric acid plant, the mixture of nitric oxide and nitrogen peroxide behaves as if these gases had combined to form nitrogen trioxide. In 1905 Raschig showed that the combination of nitric oxide and oxygen occurs in two distinct stages, the first giving rise to the production of gaseous nitrogen trioxide, which then undergoes further oxidation to nitrogen peroxide. Any further doubt as to the possibility of this trioxide existing in the gaseous state has been dispelled by the experiments described before the Chemical Society on November 7, by Dr. and Mrs. H. B. Baker who have demonstrated that liquid nitrogen trioxide can be converted into the gaseous state if it is dried completely. In this condition the gaseous trioxide shows no tendency to dissociate into nitric oxide and nitrogen peroxide; on the contrary, it exhibits the phenomenon of association, for the determination of its vapour density indicated the presence of more complex molecules corresponding with the double formula N_4O_6. This fact brings the substance into line with the corresponding oxide of phosphorus, which also has the double molecule P_4O_6. The liquid trioxide is green at the ordinary temperature, but becomes blue at −9° C. It remains liquid at −80° C., but solidifies to a mass of dark blue crystals in liquid air. These results furnish another interesting example of the influence of moisture on chemical change. Ordinary moist nitrogen trioxide dissociates almost completely in the gaseous state, but when thoroughly dried the substance may be vaporised without showing any sign of dissociation."—*Times Engineering Supplement*, Dec. 18, 1907. (J. A. W.)

Corrosion of Platinum Crucibles.—"Chemists are aware that employment of platinum crucibles involves precautions, without which the metal is rapidly destroyed. The author describes a new source of deterioration. When a platinum crucible is placed in the higher cone of the flame of a Bunsen burner, a part of the free hydrogen, always in the flame, is diffused through the platinum, and exerts a reducing action on the substances in the crucible. If by reduction substances can be formed which attack platinum, the latter is destroyed. Diffusion takes place with great rapidity. In less than five minutes the pressure increases 1½ mm. of mercury inside a platinum vessel in which a vacuum has been made. Removing the recipient from the flame, the vacuum is immediately re-established, and when placed in the lower cone of the flame for two minutes the pressure rises 4 5 mm., and the bottom becomes coated with soot. There is no diffusion if the bottom of the crucible is placed 2 cm. above the exterior cone of the flame. If a substance reduced by hydrogen at the temperature of the experiment is in the crucible, hydrogen again enters gradually as it is consumed for reduction. Iron oxide (4 gm.) is reduced to metallic iron, which renders a very thick crucible useless in five hours' heating. Magnesium sulphate is reduced to a sulphide, and sodium sulphate to the state of sulphite. Finally, magnesium pyrophosphate (produced by calcination of ammonium magnesium phosphate in the quantitative analysis of phosphates) is reduced, producing phosphorus, which distils and attacks the less hot parts of the crucible. In all these cases, when a high temperature is required, heating in an electric resistance furnace should be preferred to the use of a Bunsen burner."—M. Geibel, *Zeitschrift für angewandte Chemie.*—*London Mining Journal*, Jan. 4, 1908, p. 5. (A. R.)

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* See Proceedings, vol. ii., July, 1898, pp. 372 and 803; also this *Journal*, vol. vi., March, 1906, p. 270.

METALLURGY.

SLIME TREATMENT AT KALGOORLIE. — "At the Associated Northern mine the ore is of a schistose nature, with quartz and calcite; the valuable ore carries tellurides and iron pyrite, there being very little free gold.

The treatment consists of breaking the ore with a No. 5 Gates crusher, grinding through a 27 mesh screen in three No. 5 Krupp ball mills, roasting in six Merton furnaces, mixing with weak cyanide solution, grinding to slime, and amalgamating the coarse gold in eight Forwood-Down (improved Wheeler) 5 ft. pans, settling the slime in V-shaped boxes, agitation of pulp in five vats 22 by 6 ft., filter-press treatment in three Dehne presses, and disposal of residue by belt conveyor.

The following is the treatment in detail of the pan product. Average screen tests of this are as follows:—

Mesh.	Percentage.
On 30	Nil
40	Nil
60	0·5
80	2·5
100	4·3
150	4·9
Through 150	87·5

The slime from the settlers is run into the agitators with a consistence of about 1 to 1, it taking between 7 and 8 hours to fill one vat holding, say, 36 tons of dry slime. The cyanide solution is made up to 0·07%, and the consumption averages 1 lb. per ton treated. Agitation is continued as long as possible, but after 16 hours the pulp may be taken for filter-press treatment. For filling the press a Pearn pump having three plungers, 12 by 10 in., running at 20 rev. per min. is used. This is a powerful pump, and with good thick pulp it will fill a press in 10 minutes, lifting in this time about 10 tons of pulp, and charging against a final pressure of 60 lb. per sq. in. The time taken in filling may be divided as follows:—

Pressure.	Minutes.
Up to 25 lb.	4
,, 50 ,, ...	3
,, 60 ,, ...	1
Finishing with safety valve blowing off at 60 lb.	2
	10

The presses are of the well-known Dehne make with fifty 3-in. frames for the slime, the usual cloth-covered high and low pressure plates for filtering and hand-screwing gear. They hold about 4·5 tons of dry slime each. After a press is filled, the slime is washed for 25 minutes with weak cyanide solution, and a water wash of 5 minutes at 100 lb. pressure, during which time each ton of slime is washed by 2 tons of solution. The washing is done by a similar pump to that used in filling, only that it runs at 13 rev. per min. A mercury gauge is used on this pump. It sometimes happens that there is plenty of mill-water on hand, so the final water wash is dispensed with, the press then getting 30 minutes with solution. The decrease in the assay values of the solution during washing is:—

Period.	Assay.
At start of wash	$12·50
After 5 minutes	6·60
,, 10 ,, ...	1·50
,, 15 ,, ...	1·50
,, 20 ,, ...	1·00
,, 25 ,, ...	1·00
,, 30 ,, ...	0·80

After washing, the contents of the press are dried for 10 minutes with air at 80 lb. pressure. The press is then opened ready for discharging. Two men empty 11 presses per shift of 8 hours, say, 50 tons, on to a travelling horizontal belt conveyor 18 in. wide, which feeds an elevator belt inclined at 27°, in turn discharging on to a motor-driven swinging boom distributor.

The time taken in the different press operations is as follows:

	Minutes.
Filling press	10
Washing	30
Drying	10
Discharging..	30
Screwing-up, etc.	10

The rich gold solutions from the filter press, after being clarified in a small press, pass through three zinc boxes, and the gold (running about 890 fine) is recovered in the usual manner. Most of the solution used in washing the presses, passes into the mill again to be used in the pans, etc. An average of three months' costs of slime treatment are as follows:—

	Cost per ton.
Agitation and cyaniding ...	$0·34
Filter pressing	0·41
Precipitation, etc. ...	0·12
Disposal of residues ...	0·04
Total	$0·91

The mill has a capacity of about 3,700 tons of raw ore per month (say, 3,200 of roasted ore) and with another 1,300 tons (previously roasted) from the re-treatment of old residue; altogether some 4,500 tons pass through the three presses monthly.

Monthly returns average about:—

Ore milled, tons ...	3,700
Re-treatment, tons ...	1,300
Total value recovered ...	$70,500
Profit	$48,500

M. W. VON BERNEWITZ.—*Mining and Scientific Press*, Dec. 14, 1907, p. 743. (K. L. G.)

THE HARDINGE CONICAL MILL.—"As in almost any other improvement along any needed line, there is a diversity of opinion as to the best size of mill, quantity and size of pebbles or balls to be used, speed of mill, size of material to be fed, method of feeding, manner of lining and similar questions developed by the actual practical use of the mill. These are points which economic requirements, if not actual necessity, suggest. In my own case I had occasion to question the knowledge possessed by the manufacturers as to the actual work performed by the mill.

Their recommendation was generally for long mills of small diameter (4 to 5 ft. diameter by 15 to 22 ft. in length). Investigation developed the fact that they were often better salesmen than practitioners. Hence, when I had occasion to install one of my earlier tube mills, I reverted to a previous experiment with a comparatively short mill of large diameter, and finally carried the idea still further, by installing a mill 6 ft. in diameter, and only 8 ft. in length; in reference to which I wrote a short article on the action of this mill published in the journal of June 8, 1905, from which I quote the following, as it has bearing on a mill subsequently built, and mentioned in that article, the results of which are given later:

'The tube mills now being adopted are generally longer and of less diameter than the one I use for wet grinding, which is 8 ft. long by 6 ft. internal diameter, less the silex or flint-block lining, which reduces the actual internal diameter to about 5 ft. 6 in.; this mill, loaded with 2,500 lb. of flint pebbles about 2 in. in diameter, at 21 r.p.m. receives 6-mesh, ball-mill crushed material, and pulverises it to the sizes shown in the last column given herewith, at a rate of 3 tons per hour, which is really more than the capacity to which the rest of our plant is adapted. In view of the fact that 2,500 lb. of pebbles is not more than one-third of the supposed or theoretical load, the real capacity of the mill is undetermined.

	Ball Mill Product. Per cent.	Tube Mill Product. Per cent.
Through 6 mesh on 20 mesh	25·9	0·0
Through 20 mesh on 40 mesh	13·5	2·0
Through 40 mesh on 60 mesh	9·0	8·5
Through 60 mesh on 80 mesh	8·0	14·0
Through 80 mesh	40·0	75·5
	96·4	100·0

A few tons of quartzose ore gave a greater grinding capacity. Tests for consumption of pebbles showed less than 1 lb. per ton of ore treated. The wear upon the lining is also slight, compared with the results obtained. Based upon my present experience I have placed an order for another tube mill still further from the general practice, *i.e.*, 6 ft. long and 6 ft. in diameter.

Experiments and discussion are needed to determine the most effective diameter and length of mill; revolutions under different loads; size of pebbles; whether pebbles should be of uniform or different sizes; what relation size of feed may have to diameter of mill and of pebbles; what is the most efficient feed and discharge; whether the action is percussive crushing, or crushing-grinding, or a combination of both. The capacity and small load of pebbles in the mill I now have in use would suggest a percussive effect; certain it is that much less power is required to operate than when a larger load of pebbles is used, which would lessen the useful duty of the mill, and cause a needless attrition of pebbles and lining.'

As stated in the preceding, I then put in operation a mill, the internal diameter of which is 56 in. by 66 in. in length. This mill takes the product of the 'quartzose' ore above mentioned direct from a 14×27 in. roll, without screening or returning to the roll, giving the following results :—

RESULTS OF 56×66 IN. TUBE MILL.

Mill charged with 1,500 lb. of pebbles; consuming 15 h.p.; running 21 r.p.m., crushing 2 tons per hour.

Mesh.	On 10.	On 20.	On 40.	On 60.	On 80.	On 100.	Throgh 100.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Per cent
Heads	42·5	22·0	13·0	5·0	2·2	2·7	12·0
Tails	0·0	0·0	2·0	4·7	8·0	4·2	79·8

The above result was fully up to my expectations, considering the horse-power consumed and the pulp produced, when the coarse material fed to the mill was taken into consideration. It was results we were looking for, and we had no objection to their being more satisfactory than was anticipated; but I realised that we were producing a much larger relative quantity of fines or slimes than was desired for straight concentration.

For that reason I turned my attention to rectifying these undesirable accompaniments of the regulation tube mill. With this end in view (while never having been so unfortunate as to be within the mill while it was in operation) and forming my own 'theory of the action of the tube mill,' I set about to construct a mill which would do away with the necessity of retaining material in the mill already crushed to size desired, which greatly adds to the internal friction, and prevents the crushing action between the pebbles by their working in a medium of more or less thick mud already reduced as fine or finer than required. My object was to have the new material fed into comparatively clear water, thus the coarser pebbles or balls would act upon the coarser material, and not waste their efficiency and the power by grinding upon the smaller pebbles, which are gradually reduced in size by the work performed, and should be discarded or made to work on correspondingly small material.

This I accomplished by having the mill built with a cone outlet, or preferably two cones placed base to base. In my smelter practice I had encountered the difficulties realised by smelter men in the feeding of a furnace by having the larger portions of the ore always seek the lower point in the furnace through the momentum of a larger body falling through space or on an incline. My model was easily constructed from two laboratory funnels of 5 to 7 in. in diameter, ribbed on the inside. First filling one of the funnels about half full of fine dry sand to coarse gravel, joining the funnels with adhesive tape, and revolving the whole on a horizontal axis, immediately a displacement of the fine material by the coarse was observed, such as might be readily anticipated. It was not, however, easy to anticipate the result obtained when the same apparatus was held at a slight inclination from the horizontal and then revolved. Herein lies a curious experiment, and a result that will not be readily understood, although it is positively apparent to the eye; let the reader try it and explain it if he can.

The results produced will be seen to be more or less of a perpendicular stratification of the ground material; which stratification, passes up the inclined outlet of the mill, from the larger pebbles working on the coarsely fed material through the smaller pebbles working upon the already partially ground material. This stratification from a metallurgical standpoint, has such a decided sizing action, that its utility will be readily appreciated, and is the basis for patents granted and applied for. In fact, gradation of size is dependent upon the amount of feed, the inclination of the mill and the rapidity of discharge. To produce a finely ground material a larger load of pebbles would be required, and the mill set more approximately to a horizontal axis.

The results just obtained from a large mill constructed for fine grinding are illustrated in the following tabulation, which, considering the coarseness of the feed, the amount of pebbles used and the horse-power required, shows capacity difficult to understand although not so difficult to appreciate. Just what the maximum capacity of this mill will be, is at present unknown, as the present test was limited to the ability to feed the mill. The results are as follows :—

RESULT OF TEST OF CONICAL MILL.

Mill charged with 2,000 lb. of pebbles; consuming 15 h.p., running 25 r.p.m.; crushing 3 tons per hour.

Mesh.	On 10.	On 20.	On 40.	On 60.	On 80.	On 100.	Throgh 100.
	Per cent.	Per cent.	Per cent.	Per cent	Per cent.	Per cent	Per cent.
Heads	25·3	38·2	12·2	6·6	5·5	2·3	9·5
Tails	0·0	0·0	0·0	1·4	5·4	2·5	90·8

This result was obtained upon ore brought over 1,000 miles for the test, as the conditions for installing the mill were based upon the capacity, and the actual ore under consideration. The results obtained more than met the requirements. Those mills can be used singly, combined or in tandem.

I predict that it will be but a comparatively short time before stamps will be considered an obsolete method for crushing ore. It is, however, clearly evident to the metallurgist that at least a proportion of the ore fed to the long regulation tube mill of to-day must receive a crushing upon entering the first portion of the mill which would be sufficiently fine to make a satisfactory tailing. If this is the case, why necessitate this desirable product to work its way through 20 to 25 ft. of additional length of mill? The principle of reduction in a tube mill should be the same as that employed by any other system of gradual reduction, and not permit of its interfering with subsequent work to be performed by the mill.

Of course, the question will be asked, 'can the sizing desired be accomplished in a pebble mill?' It can. The reader may convince himself by trying the above mentioned experiment."—H. W. Hardinge. —*Engineering and Mining Journal*, Nov. 16, 1907, p. 925. (A. R.)

MINING.

Cast Iron Shaft Tubbing.—"It had been usual to calculate the thickness on the supposition that the compressive circumferential or hoop stress was taken entirely by the cylindrical shell. The number and dimensions of the ribs appeared to be chosen arbitrarily. An accurate theory must correctly apportion the stress between the flanges, ribs and shell. In an average segment of fairly heavy tubbing more than one-third of the area of the face of the tubbing had behind it a thickness of metal which was greater than the nominal thickness of the plate. It might frequently happen that the weight of metal having the thickness of the shell was only about one-third of the weight of the complete segment. The author was forced to the conclusion that the importance which the cylindrical shell theory had attained was quite artificial. The great sources of strength of tubbing lay not in the shell, but in the flanges, ribs and brackets. For rapidly determining the dimensions of cast iron tubbing for any given external pressure and diameter of shaft, he recommended the adoption of (*a*) if the stiffening rings are to be closely spaced, first the formula

$$t = kr^{2}\sqrt{lpF},$$

where t = the thickness, and F the factor of safety, taking various values of k for different proportions of w and l, the width of stiffening ring and the vertical distance between the rings respectively; afterwards testing the result by the plate formula

$$f = \frac{1}{2}\frac{pa^4b^2}{2t^2(a^4+b^4)}$$

where a, b and t are respectively the length, breadth and thickness of the plate in inches, a being greater than b, and f the greatest stress in pounds per square inch; (*b*) if the rings are far apart, use the plate formula first and then find the size of the stiffeners by the ring formula. The writer, in speaking of the pressure, said that experience with retaining walls appeared to show that Prof. Rankine's theory considerably overestimated the magnitude of the pressure. As a rough rule, it might be taken that the pressure in pounds per square inch rarely, if ever, exceeded one-half of the depth of the shaft in feet." —John Morrow.—*Colliery Guardian*, Oct. 18, 1907, p. 718. (A. R.)

Addendum to Paper on Earth Temperatures on Witwatersrand Goldfields.*—In the paper which I read before the Institution on this subject, a point in the discussion was left indeterminate, viz., the mean earth temperature at the surface in the vicinity of the Rand. A recent examination of the Report of the Transvaal Meteorological Department for 1905 has furnished me with the final figures required on this point. The conclusions arrived at, taken from a mass of detailed records, are as follows:

Mean average air temperature, taken in hourly records throughout the year, 58·2° F. Another result, arrived at by a different method of calculation, 59° F. The latter figure may be taken as more generally representative of the true conditions in the vicinity of my experiments.*

The mean earth temperatures, taken over a period of eight months, are as follows:—

1 ft. below the surface	...	64·92° F.
2 ,, ,,	...	67·987° F.
4 ,, ,,	...	67·837° F.
8 ,, ,,	...	67·63° F.

My records indicated an earth temperature at the surface of 63·95° F. In the discussion which followed the paper, in the absence of definite data on the surface temperatures, a doubt was expressed as to the accuracy of the estimate, as this amount was considered abnormally high. From the above records now quoted, it is seen that the more permanent surface earth temperatures are some 3° F. still higher than my original estimate. These results will doubtless account for the line of temperature shown on the chart in my paper in excess of the mean average, and they go to prove, that the temperatures taken at different depths have only a relative value within close limits of distance.

If any close analogy can be made between earth temperatures at the surface and those taken at depth, then any application of these official results to my records will tend to reduce still further the ratio of increase of temperature with depth in the locality under discussion, and will emphasise still more strongly the conclusion that no universal rule can be laid down for a general average rate of increase of temperature with depth, but that the ratio for each district must be estimated from purely local observation."—Hugh F. Marriott. —*The Institution of Mining and Metallurgy*, Bulletin No. 43, April 2, 1908. (J. M.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

A Treatise on Chemisty. By Roscoe and Schorlemmer. Vol. II.—The Metals. 4th Edition. Revised by Sir H. E. Roscoe and Dr. A. Harden. 30s. (London: Macmillan & Co., Ltd.)

"Sentimental reasons alone would justify a careful review of the new edition of this work, whose value has been so fully recognised since the first edition appeared, just thirty years ago, and whose pages have been so much consulted by most chemists and many metallurgists since it was first issued. One can scarcely expect a new edition of any treatise to possess the same freedom from out-of-date matter as an entirely new work should show, and the authors are to be congratulated on having brought the subject fairly into line with recent discoveries without destroying the old arrangement, with which we are so familiar. Many of the special portions have been re-written, an excellent section on radio-activity has been added, and the parts relating to alloys and crystallography have been most efficiently dealt with. Probably no work on the metals is so rich in references to early researches and ideas, and it is to be regretted that, although, as already stated, much of the matter is brought fully up to date, much has been left in which should be eliminated or replaced, and many inaccuracies have been left uncorrected.

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* See this *Journal*, vol. vi., 1906, p. 389.

* Since the Observatory is more than 300 ft. above the Rand, the mean average air temperature of the Rand is probably 62° or 63° F.—Ed. Comm.

Descriptions of obsolete processes and illustrations of plant which has been abandoned for generations are by no means infrequent, and although of some interest, are not sufficiently differentiated from descriptions of modern practice to be of benefit to the layman. In many cases the processes actually in present use are totally ignored.

The magnetic separation of the ores, now of such general use, is not even mentioned, nor has the writer seen any reference to the extremely important subject of magnetic separation throughout the book.

The subject of lead smelting is very cavalierly treated, and one might search the civilised world in vain in the hope of seeing some of the processes which are described as though in actual use.

The section on gold, although by no means short, leaves much to be desired. The assay of gold ores rightly receives but little attention, but as the method described on p. 517 as being 'usual' has probably never been employed by an assayer, it would have been better to omit it entirely.

Altogether, this work may be strongly recommended to the metallurgists as one of the best which has been published on the chemistry of the metals, but not to the chemist as a guide, or even as an introduction, to metallurgical practice."—*London Mining Journal*, Feb. 15, 1908, p. 194. (A. R.)

The Metallurgy of Steel. By F. W. Harbord, Assoc.R.S.M., F.I.C. 8¾×6 in. 3rd Edition. Price 25s. (London: Chas. Griffin & Co., 1907.)

"This book, of nearly 800 pp., is a veritable encyclopaedia of knowledge on all that appertains to steel, its manufacture, treatment, chemistry, its special alloys, its testing—in fact everything that the engineer or metallurgist must know. It is well illustrated with furnaces, rolling mills, and other machinery and general layouts. Every man interested in steel should possess this book."—*Cassier's Magazine*, Feb., 1908, p. 501. (A. R.)

Engine Room Chemistry. By A. H. Gill. 6¾× 4½ in. $1. (London and New York: The Hill Publishing Co., 1907.)

"The substance of this excellent little book appeared in our contemporary, *Power*, in a series of articles intended for enginemen, and to enable them to gain a familiarity with fuel, water, and oil, with which he must deal. The author gives a preliminary chapter in explanation of the elements of chemical and physical science. This is well done, and indeed the whole book should prove most useful to any man having to do with steam-power plants and wishing to understand the chemistry of combustion, of water softening, and so on."—*Cassier's Magazine*, Feb., 1908, p. 501. (A. R.)

The Miner's Geology and Prospector's Guide. By George A. Corder. 5s. (London: E. and F. N. Spon, Ltd.)

"Many attempts have been made to combine in a brief space those scientific principles and formulæ which are essential to the successful prosecution of mining work. Of these the most valuable represent, like the work before us, the embodied results of actual experience in the field. Mr. Corder has carried out a large amount of prospecting work over thousands of miles of territory in several continents. In addition to furnishing an exposition in little of such geological, mechanical, and mathematical matters as seem indispensable to the prospector, the book contains a large number of hints on practical questions, which, while they are almost certain to confront the actual prospector in the field, can hardly be expected to occur to the academic scientist. This is the consideration which lends such great value to essentially practical books, and befits them to occupy a useful niche in the knapsack of the traveller with mining ends in view. The book contains nothing which the miner and prospector will not be the better for having learned by heart, and represents the highest measure of ingenuity and experience in the attainment of the multum in parvo."—*London Mining Journal*, Feb. 15, 1908, p. 194. (A. R.)

Note on a Process for Extracting Gold from Aurocyanide Solutions. By P. de Wilde, Hon. Professor at the University, Brussels.

This pamphlet, which has suffered somewhat severely in translation, communicates Prof. de Wilde's new discovery with reference to the precipitation of gold from cyanide solutions by means of cuprous salts. As a reference to the back numbers of this *Journal*[*] will show, the method, which the learned professor first originated, demanded an excess of cuprous chloride for the total precipitation of the gold. He has now discovered, to quote from the pamphlet, "by acidulating the solutions aurous cyanide and cyanhydric acid are set at liberty. Very small quantities of cuprous cyanide are sufficient to absorb and fix by molecular attraction the whole quantity of aurous cyanide. By treating this precipitate with hydrochloric acid the cuprous cyanide is dissolved; the aurous cyanide remains free and by heating and melting yields pure gold without loss. It is obvious that this method is a great improvement upon the old plan, and it is possible that the idea would repay investigation. The professor's experiments seem to have been performed with fairly pure solutions, and under these circumstances complete extraction of the gold was attained. The presence in our working solutions of thiocyanates, ferrocyanides and thiosulphates would possibly have a modifying influence, and anything far short of complete precipitation would put the new process completely out of court on these fields where nothing more than a trace of gold[†] is allowed to leave the zinc boxes, and the process in use is almost automatic in its working. Prof. de Wilde's new process is admitted to be too expensive with slimes solutions because of the necessity for rendering acid such alkaline material and the formation of insoluble lime salts thereby caused. The chief difficulties unattacked in this pamphlet (besides the questions of capital expenditure for new plant and the necessity for a separate process for slimes solution) are involved in the assumption made that loss of cyanide is entirely caused by the zinc precipitation methods in vogue, and that any method of precipitation however perfect should be able to increase the extraction by so much as $\frac{1}{4}$ dwt. per ton of sand. It is now admitted that practically no loss of cyanogen[‡] takes place in the zinc boxes, and that the loss of dissolved gold in a modern plant could hardly amount to more than $\frac{1}{40}$ dwt.[**] per ton of sands, and even this small amount would be scarcely affected by absolute perfection in the precipitation department.—(H. A. W.)

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[*] Vol. ii., p. 337.

[†] Vol. vii., p. 410.

[‡] Vol. iv., p. 637.

[**] Vol. vii., pp. 243 and 411

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 115/08. Cyril Causton (1), William M. Neilson (2). Rock drill. 1.4.08.

(P.) 116/08. William Horton Fletcher. Improvements in mills or apparatus for crushing or pulverising ores, minerals or the like. 1.4.08.

(P.) 117/08. Thomas Maxhay Hack. Improvements in jockeys or rope grips for use in mechanical haulage arrangements. 1.4.08.

(P.) 118/08. James Patrick Holmes. Improvements in and relating to reversible turbines, rotary engines or motors. 1.4.08.

(P.) 119/08. George Ridgway. Improvements in filtering machines. 2.4.08.

(C.) 120/08. Wilbur Alson Hendryx. Combination agitator and decanting filter. 3.4.08.

(P.) 122/08. William Thomas. Improvements in means for elevating granular, material, liquids or mixtures of liquids and solids. 4.4.08.

(C.) 124/08. William Cumming. Guide block. 6.4.08.

(P.) 125/08. Michael Neilson. Improvements relating to the dumping of trucks or like vehicles on similar depositing sites. 6.4.08.

(P.) 126/08. Charles Hansen. Improvements in the safety brake arrangements for mine cages and other hoisting apparatus. 6.4.08.

(C.) 127/08. Herbert Lansing Merrick. Integrating apparatus. 7.4.08.

(C.) 128/08. Bruce Edward Tennant. Improvements in assay furnaces. 21.11.07.

(P.) 129/08. Alexander McNamara. Improvements in means for connecting the tube rod to the cord barrel and sediment drill of rotary boring apparatus. 8.4.08.

(P.) 130/08. Ivor Bevan (1), Frederic Anderson (2). Improvements in straightway valves. 10/4/08.

(C.) 131/08. Donald Donald. Improvements in wire strainers. 10.4.08.

(P.) 132/08. Isaac Frames Taylor (1), Samuel Pick (2). Improvements in cranes, fire escapes and similar apparatus. 10.4.08.

(P.) 133/08. John Hudson Hughes. An improved machine operated by compressed air for drilling holes in rock or other material. 10/4/08.

(C.) 135/08. Frederick Charles Lynde (2), Clement Vincent Haworth (2). Improvements in and relating to apparatus for the production of gas. 10.4.08.

(C.) 136/08. Frederick Capel Brown. An improved lining for tube mills and for tube mills and ball mills and similar grinding and pulverising machines. 10/4/08.

(P.) 137/08. John Grant (1), Alexander Will (2). Improved air or steam worked rock drill. 10.4.08.

(P.) 138/08. Henry Percy Forster. Improvements in wheels for motor and other vehicles. 11.4.08.

(P.) 139/08. Montague Moore (1), Thomas James Heskett (2). Improvements in the process of and apparatus for reducing iron ore and subsequently treating same for the manufacture of wrought iron and steel. 13/4/08.

(P.) 140/08. Frederick Daniel Melvill. Automatic door-brake. 13/4/08.

(C.) 141/08. William Henry Appleby. Improvements in and relating to automatic couplings for all rolling stock. 13.4.08.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

ARKELL, D. J., *l/o* Johannesburg; Kimberley Reef Syndicate, Mazoe, Rhodesia.
BROOM, W., *l/o* Johannesburg; Molyneux Mine, Kraal Station,
BROWN, G. D., *l/o* Horncastle; Hotel Victoria, Johannesburg.
CRIBBES, G. C., *to* Robinson Randfontein G. M. Co., Ltd., Randfontein.
CROSSE, A. F., *l/o* Pilgrims' Rest; P. O. Box 598, Johannesburg.
ERSKINE, C. H., *l/o* Johannesburg; Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston.
JEFFERY, A. R., *l/o* Que Que; Francistown, B.B.
FERGUSON, A. M., *to* 20a, Johannes Street, Troyeville.
FITCH, FRANK, *l/o* Arizona; c/o Ironton School, Denver, Colorado, U.S.A.
GRAHAM, W. H., *l/o* Francistown; Parkfield, New Ferry, Cheshire, England.
HARTLEY, A. H., *l/o* Geldenhuis; P. O. Box 1096, Johannesburg.
ISAACS, R. McI., *l/o* Rhodesia; 63, Charlton Road, Blackheath, London, S.E.
JOHNS, JAMES, *to* Ginsberg G. M. Co., Ltd., P. O. Box 16, Knights.
JOHNSTON, JAS., *l/o* New York; Chas. Butters & Co., 54, New Broad Street, London, E.C.
LATHBURY, F. H., *l/o* Johannesburg; Antelope Mine, c/o R.E.D. Co., Bulawayo, Rhodesia.
MCCORMICK, W. E., *l/o* Denver; South Rand G. M. Co., Ltd., P. O. Box 102, Fordsburg.
MOORE, K. BYRON, *l/o* Australia; Alaska Perserverance Mining Co., Juneau, Alaska.
NEWTON, S., *l/o* Fordsburg; 66, St. Philips Road, Norwich, England.
NEWSON, J. A., *l/o* Klerksdorp; P. O. Box 250, Potchefstroom.
QUIN, S. C., *l/o* Cleveland; P. O. Box 6, Krugersdorp.
READING, R. W., *Journals to* c/o R. L. Reading, Redding, Shasta County, Cal., U.S.A.
SCOTT, A. D., *l/o* Cleveland; Durban Roodepoort Deep, Ltd., P. O. Box 110, Roodepoort.
STROUT, E. A., *l/o* Roodepoort; Witwatersrand Deep, Ltd., P. O. Box 5, Knights.
THOMAS, Jr., *to* "Manzar," Barncoose, Redruth, Cornwall, England.
TRUSCOTT, S. J., *l/o* Sumatra; c/o Messrs. Aschoff & Co., 3, Market Buildings, Mincing Lane, London, E.C.
WILLIAMS, G. W., *l/o* W. Australia; c/o *Australian Mining Journal*, Queen Street, Melbourne, Australia.

THE JOURNAL
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VOL. VIII. MAY, 1908. No. 11.

Proceedings
AT
Ordinary General Meeting, May 16, 1908.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, May 16th, Prof. J. Yates (President), in the the chair. There were also present :—

51 Members : Messrs. F. F. Alexander, R. G. Bevington, W. R. Dowling, A. Heymann, A. McA. Johnston, A. Richardson, G. O. Smart, Prof. G. H. Stanley, A. Whitby, H A. White, Prof. J. A. Wilkinson, W. A. Caldecott, S. Beaton. W. Beaver, G. D. Brown, E. H. Croghan, A. F. Crosse, Hon. W. Dalrymple, M.L.C., G. A. Darling, P. L. Edwards, A. D. Gilmore, J. Gray, B. J. Hastings, M. Inglis, J. I'Ons, H. R. Jolly, J. A. Jones, G. A. Lawson, Hy. Lea, Jas. Lea, W. P. O. Macqueen, G. Melvill, J. E. Metcalf, J. T. Milligan, A. Mitchell, P. Morrisby, B. P. Mosenthal, W. Nicklin, W. J. R. North, L. J. Robinson, W. H. Roe, O. D. Ross, C. E. Rusden, C. B. Saner, H. A. Scarf, A. L. Spoor, H. Taylor, J. E. Thomas, J. F. Walker and J. P. Ward.

14 Associates : Messrs. S. J. Cameron, J. Chilton, C. L. Dewar, A. L. Edwards, B. Ellis, J. H. Harris, C. B. Hilliard, W. H. Jane, H. Rusden. A. Thomas, W. E. Thorpe, W. Waters, L. J. Wilmoth and J. A. Woodburn.

10 Visitors and Fred. Rowland. Secretary.

The minutes of the previous monthly meeting as published in the *Journal*, were confirmed.

NEW MEMBERS.

Messrs. G. Melvill and P. Morrisby were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that the candidates for membership had been duly elected, as follows :—

HENDERSON, JOHN MCCLELLAND. P. O. Box 1146, Johannesburg. Mining Engineer.
JOHNSON, HUGH MICHELL LUTTMANS. Rand Club, Johannesburg. Mining Engineer.
MATHER, THOMAS W., South American Development Co., P. O. Box 336, Guayaquil, Ecuador.
SMITH, EDWARD, French Rand G. M. Co., Ltd., P. O. Box 92, Luipaardsvlei. Amalgamator.
SUART, ARTHUR B., Johnson & Son's Smelting Works, Ltd., Finsbury, London, E.C. Bullion Refiner.
THOMAS, ABRAM, Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Smelter.
(Transfer from Associate Roll.)
WARWICK, D.S.O., JOHN ABRAHAM, Bushtick Mines, Ltd., per private bag, Bushtick Siding, via Bulawayo, Rhodesia. Mining Engineer and Mine Owner.
WILLIS, HARRY THOMAS, Champaign, Illinois, U.S.A. Mining and Metallurgical Engineer.

The Secretary announced that the following gentlemen had been admitted as Associates by the Council since the last general meeting :

GEPP, HERBERT WILLIAM, c/o de Bavay Treatment Co., Broken Hill, New South Wales. Chemist.
TAYLER, WALKER ALFRED CHARLES, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

The following were then proposed and unanimously elected scrutineers for the forthcoming ballot for officers at the annual election :—

Messrs. Gray, W. E. Thorpe, Morrisby, J. Lea and A. L. Edwards.

The President : I do not know whether you have noticed the omission of one name from the ballot paper for the new members up for election to-night—the name of Mr. Lambert. I am very sorry to say that Mr. Lambert died a few days ago at Lydenburg from a chill, followed by pneumonia, and the end came in two or three days. Under the sad circumstances, I move that we send a letter of condolence to his relatives.

The motion was adopted unanimously in silence.

The President : The Council, at its last meeting, unanimously decided to make Prof. Hilary Bauerman, A.R.S.M., F.G.S., etc., an honorary member of our Society. He is an honorary member of the Institution of Mining and Metallurgy, London, and, in 1906, was awarded the gold medal of that Institution in recognition of his services to metallurgy. He is one of the veterans of science, and the Council feels sure that in conferring the honorary membership upon him as a mark of our appreciation of his work, it has your unanimous and warm approval.

The Secretary will now read a letter received from the Kleinfontein mine, with reference to a certain accident which occurred recently in the cyanide works.

The Secretary: This letter was received in reply to an enquiry suggested by a member of Council, as to the cause of the accident.

Kleinfontein, Box 88, Benoni,
April 30th, 1908.

"In reply to your enquiry, I have pleasure in sending you the following account.

This accident happened in one of two 30-ton vats, which are used for receiving the fairly clear liquid decanted from the acid treatment tank. The liquid is allowed to settle in these tanks for about a fortnight before being decanted off and run to waste. The small amount of slime which settles is occasionally cleaned up, and on the 20th April, when the accident occurred, one of these vats had already been cleaned, and a boy who had been engaged in cleaning the first tank had started to clean up the second: things went quite smoothly for about half an hour until another boy was sent down to help, immediately after which the accident happened. The boy who was first in the tank feeling himself affected by fumes succeeded in getting out, but the other, who was following, showed signs of collapsing, and a white man and three natives went to his assistance. All the natives were overpowered by the fumes, and two of the native rescuers seem to have met with instant death. These two, with the boy they were trying to save, fell to the bottom of the vat. The white man, although partially unconscious, succeeded in getting away and in dragging with him the third native rescuer, who was brought round by means of ammonia and the application of cold water to his back. After the accident happened there was a strong smell of hydrocyanic acid in the vat, and there can be little doubt, that in some way the second boy had introduced cyanide into the vat, which was decomposed by the slightly acid sludge with the liberation of HCN. We are quite at a loss to surmise where the cyanide came from. As the vat is situated there is no possibility of leakage of solution into it, and the buckets used were specially marked buckets used only for this purpose. The fact that one boy had already been half an hour in the vat without feeling any bad effects would indicate that the cyanide must have been introduced at the time the second boy went into the vat.

The accident was an unusual and unexpected one, as under ordinary circumstances there should be no possibility of there being any appreciable quantity of poisonous gases in the vat in which it happened. The only precaution against the repetition of such an accident which suggests itself to me, in addition to the obvious one of exercising the greatest care in the handling and storing of cyanide, is that of having air pipes led to the vats so that in the event of the accidental liberation of poisonous gases these could be rapidly diluted to such an extent as to be rendered harmless. Since the accident happened we have had this done at the Kleinfontein cyanide works, so that we can blow air into the vats when men are at work in them."

J. K. Erskine, Cyanide Manager.

The President: I have just had placed before me a letter, addressed to the Secretary, which I will read.

METALLURGICAL TRIALS' COMMITTEE.

Dear Sir,

I beg to inform you that a committee has been formed of technical representatives of the majority of the mining corporations on the Rand, for the purpose of developing original suggestions, ideas and inventions (during the experimental stage), for improvements in the treatment of Rand ore. The following corporations are represented on this committee:—

Consolidated Gold Fields of S.A., Ltd., Messrs. H. Eckstein & Co., Rand Mines, Ltd., Farrar-Anglo-French Group, Messrs. S. Neumann & Co., General Mining and Finance Corporation, Ltd., Consolidated Mines Selection Co., Ltd., and Messrs. Lewis & Marks.

I have been instructed to request you to kindly notify the members of your Society of the formation of this committee, and of the opportunity thus afforded for bringing forward suggestions for improvements in ore treatment. The basis of remuneration or royalty payable for any suggestion experimented on will be arranged before the commencement of the trials.

Communications relating to proposed improvements should be submitted in writing to the Secretary.

J. P. Long, Secretary

I must say that this idea of forming a committee to go into the merits of new inventions, and of proposed improvements, has my hearty approval, as I feel sure it has yours. To me it does seem, that the idea of taking new notions to this committee rather than to individuals is better calculated to secure full justice and consideration for such ideas. We have, on the Rand here, often been accused of indifference to new ideas, and I remember well that Lord Selborne, in connection with the awarding of the trophy to the Gordon drill, in the recent competition, refuted that accusation. Now, I think, that you will agree with me that the idea underlying the formation of this committee is excellent, and I would remind you that all communications in connection with the committee mentioned are to be addressed to Mr. Long.

The Secretary: In connection with the annual election of Officers and the Council, the nominations must be in my hands by the 24th May, and I shall be obliged if members will send them in early.

The President: I have just been reminded that Dr. Moir is shortly leaving for home on a six months' holiday. I am sure that with me you wish him a very pleasant trip, and a safe return.

As you are aware, we visited the Nigel mine last Saturday, and I think I am right in saying that those who accompanied the excursion enjoyed themselves most thoroughly. I move that the Secretary be instructed to send a letter of thanks to Messrs. Line and Curnow for their excellent hospitality.

The motion was carried unanimously.

The Secretary: I am continually getting applications from men who want engagements as cyanide learners, assayers, etc., and I shall be glad if cyanide managers and others, who are in need of such men, will let me know, so that I can put such applicants into communication with them.

The President: I am very pleased to see that we have to-night two very old members here, Mr. J. R. Williams and Mr. Crosse. Your applause at least tells them that they are very heartily welcome. Mr Crosse has a few words to say.

Mr. A. F. Crosse: With reference to Mr. Sinclair Brown's letter, read at the March meeting of the Society, combating my statement made at the September meeting, that nitric acid was used to dissolve the copper out of the base bullion, which, together with the silver dissolved, accompanied the copper down the creek, I wish to say that I fully accept Mr. Brown's statement that the copper and silver were duly recovered and accounted for, and did not go down the creek as I somewhat jocularly remarked. But I do wish to say a few words as to the method adopted by Mr. Brown. There is no possible advantage to be gained in treating gold precipitate containing copper and silver with nitric acid, which is very expensive—especially in an out of the way district. Mr. Scrymgeour, of the Transvaal Gold Mining Estates, Ltd., Pilgrim's Rest, informs me that cupriferous precipitate is simply treated with sulphuric acid to remove the zinc, and then shipped to London, where it realises a very high value for the gold and silver contents, and a good value for the copper. The final result, from a commercial point of view, is infinitely better than the method adopted by Mr. Brown.

THE ADAIR-USHER PROCESS.

By Alfred Adair.

In response to the invitation to contribute a paper on the above, I have pleasure in submitting to the members of the Society, an account of the original experiments, and the ideas which sprang from them, judging they would prove of greater interest than a bare record of results.

The first experiments related to the use of umber in slimes treatment, and these shewed the possibility of reducing the time of treatment, which led to the Usher apparatus. The low residues obtained demonstrated the possibility of lowering sands residues by efficient tube milling, and as this material might contain a considerable amount of sand, it suggested a further advance in apparatus. Each of these subjects will be dealt with in sequence.

Umber.—This mineral is found in most countries, and consists of a mixture of oxides of iron and manganese. It is usually soft and pulverulent. The German mineralogical name is Braunstein, or Umbra. The Transvaal mineral occurs all over the dolomite areas. The largest and nearest deposit is found N. of Krugersdorp—sandwiched between the dolomite and its underlying strata. It is a sedimentary deposit but not exactly stratified. The outcrop is 15—20 or even 30 ft. wide, and it extends for many miles both E. and W. As the dolomites in this locality contain 7—8% of ferrous and manganese carbonates, this occurrence of umber is readily explained. The mineral has the character of a chemical precipitate, is in an extremely fine state of division, and contains combined water. The chief impurity is silicious matter 10% to 40%, and it rarely contains less than 30% of moisture, which it retains with great obstinacy. The colour varies from light brown, through purply shades, to a dark rich brown. The depth of colour indicates the greater preponderance of manganese dioxide, but is only an approximate guide to the percentage thereof. No two samples agree exactly in composition, not even when they are in juxtaposition. The following analysis is not an average sample, the proportions of silicious matter and manganese dioxide being high. The sample was a large one and was dried at 212° F., constant weight being obtained only after 48 hours' drying, the sample losing 31% H_2O.

	per cent.
Silicious matter	38·93
Alumina...	4·01
Manganese dioxide ...	26·40
Cobalt and nickel oxides ...	·05
Calcium carbonate ...	1·80
Magnesium ,, ...	1·47
Phosphates	traces.
Sulphates	traces.
Combined water	7·66
Ferric oxide	19·43
	99·75

Analysis of the silicious matter—

Silica	30·29
Alumina	6·42
Ferrous and manganese oxides	traces.
Lime	·23
Magnesia	·56
Alkalis and loss	1·43
	38·93

The variability in the composition of umber is shown by the following partial analysis, made in the Government Laboratories :—

	per cent.
Silica	29·54
Iron oxide	42·50
Manganese oxide	9·40
Loss on ignition	16·50
	97·94

Transvaal umber is particularly sensitive to chemical reagents, which suggests its use in other ways than the one to be described. It absorbs sulphuretted hydrogen, and especially sulphurous anhydride with great avidity, forming the dithionate of manganese in the latter case. It is a powerful deodoriser, and it also yields a very pure spiegeleisen with ease on smelting with lime.

Initial Experiments with Umber.—Solutions of ferrous sulphate were prepared, to which lime was added in excess, and then umber, or powdered pyrolusite. The solutions to which umber was added shewed a marked decrease in ferrous hydrate, while the pyrolusite tests shewed no decrease due to the reagent. Cyanide solution to which umber was added shewed no decrease in strength, but became slightly yellow, due to dissolved manganese. After some small tests on slimes, which were favourable, a test on a large scale was made by adding 1 ton to 140 tons of accumulated slimes. The result showed an extraction of 80% on first transfer, as compared with similar charges of 66%, and the loss of cyanide nil, as compared with 15% without umber. These tests were not considered as quite conclusive at the time, and therefore a rather long series of parallel experiments, 26 in all, was made, for the sake of experience, as well as for the primary object of proving the value or otherwise of using umber.

Parallel Experiments on Slimes with and without Umber.—These experiments, illustrated by the accompanying diagrams 1 to 8, were made in two small hand churns, which were placed side by side with the handles inward, so that each received the same amount of agitation. Both received exactly the same treatment, so the results are strictly comparable. Owing to the smallness of the charges—15 litres of pulp in each—it was not possible to make many estimations. The estimation of FeO in the presence of cyanide was attempted, but the results were only comparable with each other, and, being imperfect, are unfitted for publication. The estimation of very small percentages of FeO in slimes, in the presence of cyanide, ferrocyanide, ferric oxide, manganese dioxide, ferrous sulphide, lime and alkali, etc., offers quite a pretty problem in analysis. A reliable method, which could be readily performed, would be very useful.

The Assays.—The assays in many cases involved small differences, so the method is given. Each sample for undissolved gold was washed twice, each time in four gallons of water. Each pair of assays was done in duplicate and fused, scorified, and cupelled together, 4 to 5 A.T. being used for each assay, and a button of about 70 gm. lead reduced. This was scorified to 10 gm. before cupellation. If this scorification is adopted, the final results are more trustworthy.

Cyanide Estimations.—The samples were filtered off, and tested immediately, 100 c.c. usually being taken.

The results of the more interesting experiments are shown in diagrammatic form. The ordinates give percentages and the abscissæ the hours of treatment. The percentage loss of cyanide is shown by the dotted lines, and the percentage extraction of gold in full lines. The light lines refer to the unumbered charges (A) and the heavy ones to the umbered charges (B).

Diagram I.—Accumulated slimes.
Acidity ·0193% H_2SO_4. Fe as $FeSO_4$. ·0474%
Lime to each in slight excess. Umber ½%
Ratio of solution to dry slimes 1·2 to 1
Ratio of cyanide to dry slimes ·036 to 100
Strength of cyanide solution ·03%.

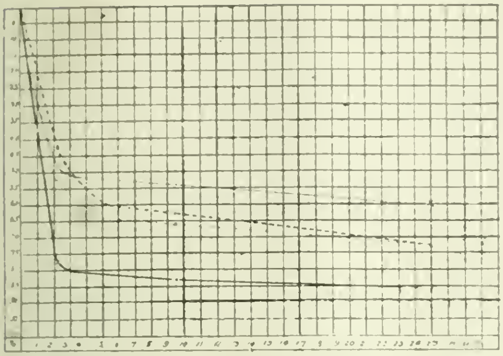

FIG. I.

Diagram II. — Accumulated slimes, same sample as I., but exposed longer to air.
Acidity............? Fe as $FeSO_4$. ·0081%
Lime to each in excess. Umber 1%
Ratio of solution to dry slimes 2 to 1
Ratio of cyanide to dry slimes ·18 to 100
Strength of cyanide solution ·045.

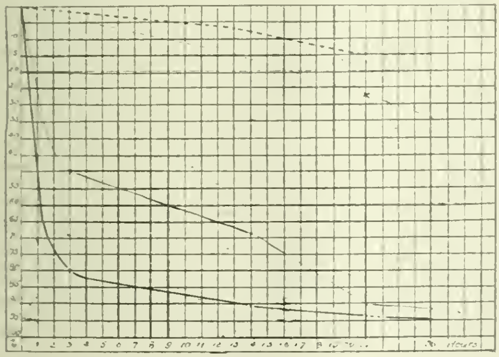

FIG. II.

The solution of the gold. up to three hours, is approximately equal in each case. notwithstanding the difference in the cyanide ratios.

Diagram III.—Accumulated slimes.
Acidity ·0594% H_2SO_4. Fe as $FeSO_4$. ·056%
Lime to each in excess. Umber 1%

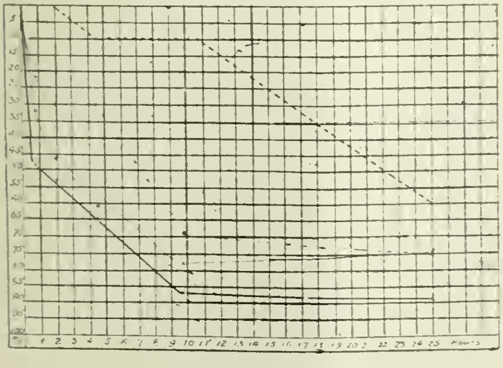

FIG. III.

Ratio of solution to dry slimes 2·27 to 1
Ratio of cyanide to dry slimes ·0454 to 100
Strength of cyanide solution ·02%.

The consumption of cyanide in A is apparently due to formation of prussiate between 5th and 9th hours more particularly.

Diagram IV.—Accumulated Slimes.
Faintly acid. Dark grey in colour. Fe as FeO ·24%
Lime to each in slight excess. Umber 1%
Ratio of solution to dry slimes 2·05 to 1
Ratio of cyanide to dry slimes ·041 to 100
Strength of cyanide solution ·02%.

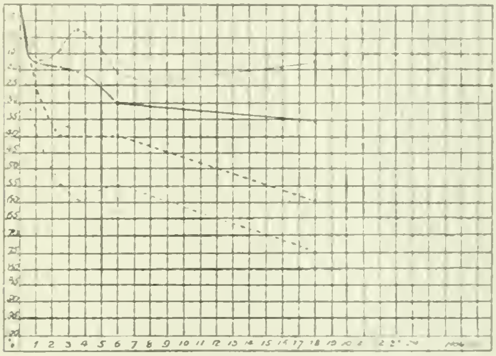

FIG. IV.

This is the most interesting experiment in the series. The amount of cyanide was far too small. but it was fortunate that it was so, as it furnishes direct proof of the reprecipitation of gold in the un-umbered charge.

Another experiment with the ratio of cyanide to dry slimes of ·14 to 100 gave the following results :—

hours.		A Cy. loss.	Gold extr.	B Cy. loss.	Gold extr.
1	...	?	3·4	0	20·38
3¾	..	25	6·23	9	25·47
4¾	...	30	12·27	15	33·02
6	...	35	13·40	17	35·06
10	...	45	13·21	25	43·40

The same sample of slimes as the above, was broken up and allowed to oxidise in the air for 14 days, the oxidation being almost perfect. Slight excess of lime was added and 1% umber.
Ratio of solution to dry slimes 1 to 1
Ratio of cyanide to dry slimes ·05 to 100.

	Extraction.
After 10 hours' treatment 1 to 1 ...	93 %
After 14 hours' treatment ...	95 %
After 25 hours' treatment ...	95½%
After 3 washes, total 4 to 1 ..	96 %
After 4 washes, total 5 to 1 ...	96¾%

Diagram V.—Current slimes neutral.
Fe as FeO. ·0137%
Lime in slight excess to each. Umber 1%
Ratio of solution to dry slimes 2·3 to 1
Ratio of cyanide to dry slimes ·069 to 100
Strength of cyanide solution ·03%.

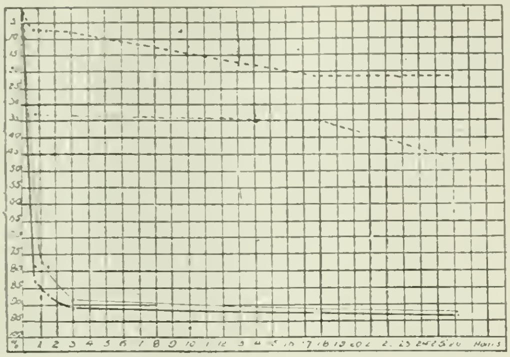

Fig. V.

Diagram VI.—Current slimes neutral.
Fe as FeO. ·0384
Lime in slight excess to each. Umber ·3%
Ratio of solution to dry slimes 2 to 1
Ratio of cyanide to dry slimes ·04 to 100
Strength of cyanide solution ·02%.

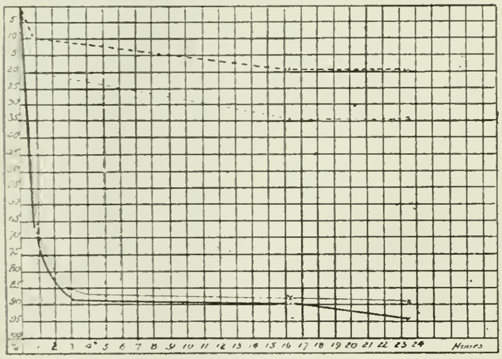

Fig. VI.

Diagram VII.—Current slimes alkaline and oxidised.
FeO traces
Lime already in excess. Umber 1%
Ratio of solution to dry slimes 2 to 1
Ratio of cyanide to dry slimes ·032 to 100
Strength of cyanide solution ·016.

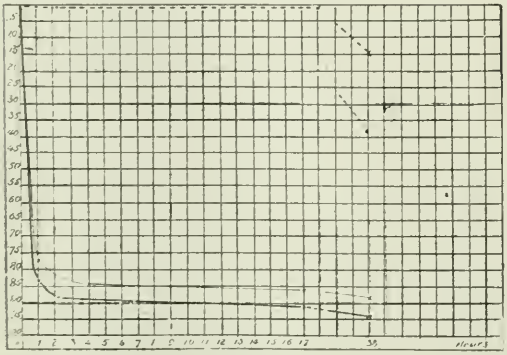

Fig. VII.

As might be anticipated, umber is most useful with the weakest solutions.

Diagram VIII.—Usher apparatus. Current slimes, neutral and oxidised.
Fe not estimated
Lime in slight excess to each. Umber $\frac{1}{2}$% to B.
Ratio of solution to dry slimes 2 to 1
Ratio of cyanide to dry slimes ·04 to 100
Strength of cyanide solution ·02%.

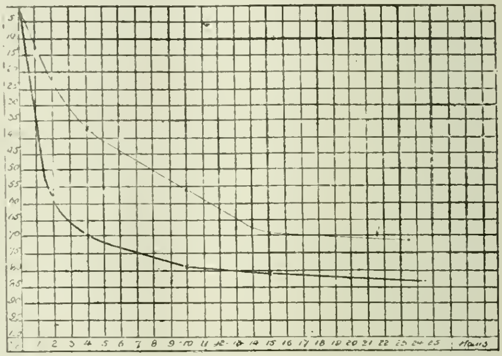

Fig. VIII.

After 15 hours' treatment washed with 4 to 1 of water.

In taking the foregoing results as a whole, whatever the reasons may be, it is clear that the umbered charges have invariably given the best results, and such has been found to be the case in every instance. The addition of umber is beneficial to the extent of from 3d. to 1s. 6d. per ton in gold and cyanide, according as slimes are low or high grade, and as the cost, including royalty, does not exceed 2d. per ton, its adoption will yield a greater profit and add to the security of the process. It is especially useful in reducing the time of treatment, which is now claiming the attention of those who desire to be abreast of the best practice. In attempting an explanation of the reasons for the action of umber, much more research work requires to be done. It has been generally observed, that ferrous compounds always have a detrimental effect on extraction by cyanide, and that aeration, or the addition of oxidising agents, such as $KMnO_4$ and Na_2O_2, distinctly improves the process, but at very considerable cost. The ferrous salts in the presence of lime and alkali, form ferrocyanides; in sand solutions of ·2 KCN or thereabouts, this is not serious, but with slimes solutions one-tenth of that strength and the iron compounds in a much finer state of division, it is no uncommon experience to find the whole of the cyanide converted, and the gold accordingly mostly re precipitated, or not dissolved, and a second addition of cyanide becomes necessary.

The ferrous compounds existing in alkaline slimes are the hydrate or the sulphide; the latter is usually small in amount, but present in all the samples, even of current slimes that I have examined. It is the greatest foe the cyanider has to deal with, as shown in experiment 4, where there is a very clear indication of the reprecipitation of gold. These particular experiments were carried out on pure black slimes, a very small proportion of which usually enters the treatment vats. Here umber shows a distinct improvement and its capacity to deal with the usual smaller amounts of sulphide. On ferrous hydrate in the presence of lime the action of umber is to oxidise it to the ferric compound, being itself reduced to Mn_2O_3, which does not affect the process.

If a charge be thoroughly aerated, it might be expected perhaps that the addition of umber would be without any effect, but such is not the case. Several of the experimental charges were thoroughly aerated, No. 8, for example, and yet the umbered charge not only gave a better extraction, but also conserved the cyanide better. The explanation offered of this effect is, that on the addition of cyanide to an umbered charge, a portion of the manganese dioxide is instantly dissolved. If a 1% cyanide solution be shaken up with umber, it instantly becomes yellow, forming a soluble double compound, which instantly oxidises ferrous salts. This double cyanide is not a manganic but a manganous, or perhaps sesqui-cyanide compound.

Manganese Double Cyanide.—I propose to add a few remarks on the manganese double cyanides, as they are of interest, yielding several compounds. By excluding air, a colourless solution is obtained from cyanide and a manganous salt, which on partial aeration turns pink, on further oxidation brown, and finally yellow. On oxidation with chlorine, bromine, or Na_2O_2, it becomes reddish-brown. On acidification, the colourless solution gives a whitish precipitate with ferrous sulphate, and each of the others blue. The last solution only gives a brown colouration with ferric salts, and all the others blue: it is apparently a mangani-cyanide, corresponding to ferricyanide.

A further examination of these compounds would yield some interesting chemical results, and possibly modify our present conception of the formulæ of double cyanides.

THE USHER WASHING APPARATUS.

From the foregoing experiments it is clear, that the time of treatment for slimes can be considerably reduced; but by the old decantation process, it is not possible to do so, hence the necessity for a more rapid process, which has been realised in the Usher plant. This modification of the decantation process not only shortens the time of treatment, but what is perhaps of more importance, it permits of more perfect washing, and therefore gives lower residues.

In its simplest form, the plant consists of a pipe, leading from the top to the bottom of the ordinary tank, and there branching out into numerous small perforated pipes, which should spray every part of the bottom of the vat. These pipes may be 1 in. in diameter and 2 ft. apart. The holes are so placed, that a slight rotary motion is given to the pulp.

The original form of the apparatus was a central down pipe, 9 in. in diameter, with four arms of 5 in. pipe flattened, with 50 $\frac{1}{2}$-in. holes distributed along the arm, according to the sectional areas of the vat, and driven by a rope, which passed round a large pulley, fixed to the arms and supported by tie rods. The rope passed round pulleys at the required height, and over a driving pulley placed at the top of the vat.

The simpler arrangement, being cheaper, has been the form usually adopted, but misses one or two advantages of the other system, viz., saving in power, and more efficient treatment. As the object of this paper is to show the experimental side of the question, I leave the large scale experiments to be described by those who have personally conducted them, as I am only responsible for the laboratory experiments.

The first Usher experiment was made in a tub of about 60 litres capacity. The apparatus consisted of a $\frac{3}{4}$ in. central tube with two horizontal arms, perforated with $\frac{1}{16}$ in. holes, which were joined to the upright pipes, and extended to the sides at the bottom of the tub, and were revolved at a very slow rate for the first 24 hours, and more rapidly for the remaining 12 hours; 30 lb. of slimes with 30 lb. of ·02 % cyanide solution were then charged into the tub and the arms revolved. Cyanide solution ·02 % was then passed through for 36 hours. The results were as follows:—

Original slimes ...	3·34 dwt.
Washed residue ..	·5 ,,
Unwashed residue ...	1·49 ,,

The suspension in this experiment was imperfect, and led to channelling, but the main feature aimed at was to get clear solution at the top, while the arms were in motion and solution in circulation; in this we were quite successful. It was evident that the general idea was right, and with some modification the extraction could be improved.

Usher Experiment, No. 2.—The same stirrers as in last experiment were used, but stirrers were soldered to the arms.

Charge 1, solution ·027% KCy to 1 of current slimes
½% lime and ½% umber.
Time in circulation, 23¾ hours, ½ to 1 of ·025 solution added—
Washing, 3 to 1 of dry slimes in 8 hours.

Results—

Original slimes...	...	...	3·37 dwt.
Washed residue	¾ hour	·94	72% extr.
Washed residue	23¾ hour	·12	96·3% ,,
Washed residue	final	·10	97·0% ,,
Unwashed residue	final	·18	94·6% ,,
First solution drawn			1·115 dwt. per ton
Last solution drawn			·18 dwt. per ton

Total solution, 4½ to 1.

In this case complete success was achieved, as one may say, at a single bound.

Usher Experiment, No. 3.—Similar to No. 2.

Usher Experiment, No. 4.—

Charge 1 of ·025 solution to 1 of dry slimes
¼% lime and ¼% umber
1 of solution, ·025% KCy added in 10 hours } 30¼
4 of wash water ,, 20¼ ,, } hours
Total solution and wash, 6 to 1
Solution drawn off, 4 to 1.

Cyanide in first solution drawn	...	·024
Cyanide in last solution drawn	...	·003
Alkalinity in first solution drawn	...	·02
Alkalinity in last solution drawn	...	·002
Gold in decanted solution, 4 to 1	...	89%

Assays—

Original charge	...	3·35 dwt.		
Unwashed residue	28¼ hours	·18	,,	94 6%
Washed residue	28¼ hours	·10	,,	97·0
Unwashed residue	30¼ hours	·10	,,	97·0

The value of the final wash is very marked bringing the actual extraction up to 97%. The next experiment suggests further possibilities.

Usher Experiment, No. 8.—The charge was made up to 3·4 of solution to 1 of dry slimes, ½% lime, ½% umber, and at once set in agitation. After a short time the solution began to clear, and as it was decanted off, it was again put through the pulp continuously for 18 hours.

The charge was then washed with 5 and 3 to 1 of dry slimes in 7 hours. Total solution 8·7 to 1.

Assays—

Original slimes (current)		2·0 dwt.	
Washed residue	12 hours	·20 ,,	90·0% extr.
,,	18 ,,	·12 ,,	94·0 ,,
,,	25½ ,,	·10 ,,	95·0 ,,
,,	30½ ,,	·025,,	98·75 ,,
,,	32½ ,,	·02 ,,	99·0 ,,
Unwashed residue	30½ ,,	·08 ,,	96·0 ,,
,,	32½ ,,	·03 ,,	98·5 ,,

In all, some 12 experiments were completed. The following notes have been deduced from the experiments :—

1. ½ solution of ·04 gives much quicker results than 3 to 1 of ·02 cyanide solution.

2. The time of treatment is reducible to 24 hours, if after withdrawing the decanted solution the balance is discharged to a dam, subsequently recovering the bulk of solution.

3. The less the amount of original solution, the quicker and more satisfactory is the recovery of the gold.

4. The rate of washing depends on the slimes, and the bulk of solution in the vat, also on alkalinity and temperature ; in the Usher process the alkalinity is of more moment than the temperature. Other things being equal, the rate of washing an original 2 of solution to 1 of dry slimes pulp :—

$$\frac{\text{Units (in tons) of wash per hour}}{7} = \text{Rate of tons}$$

of wash solulution per hours at 3 to 1.

$$\frac{\text{Units (in tons) of solution}}{4} = \text{Rate cf washing in}$$

tons per hour solution 4 to 1.

$$\frac{\text{Units (in tons) of solution}}{2} = \text{Rate of washing in tons per hour.}$$

Thus, if the bulk of pulp contained 400 tons solution to 100 tons of slimes the rate of washing would be 50 tons per hour. The efficiency of the treatment is affected by the temperature. To obtain proper replacement, the temperature of the feed should be some degrees below that of the temperature of the vat.

Practical Results.—The figures for the practical work are not in my possession, but I understand that the general result of the partial adaptation of the process shows a distinct gain in time and lower residues, at a less cost.

New Continuous Process, all Sliming.—In our great industry there can be no finality to any process, and one desideratum has always been simplicity, and ever more simplicity, especially in the elimination of labour wherever possible. To achieve this *a continuous process* eliminating Kafir labour, gives the best hopes of success. Several earnest trials have been made, both here as well as in Australia and America. I need only quote the E. T. Rand and Clancy processes as proof.

The method which I am here bringing forward, differs from any other in some essential particulars. As it has been shown in the experiments, and in actual practice, that a much better recovery is obtainable from slimes than from sands, it follows if the cost of reduction of sands to fine sands or slimes can be effected for the difference in cost between the two processes, then the extra gain in gold would be just so much extra profit.

My proposition is to put in more tube mills, grind finer, and treat the resulting pulp in one plant instead of two. The present difference in working costs between the sands plants and the slimes plants, is approximately 9d. to 1s. per ton, due largely to native labour. The present price of tube milling, reducing the sands to 90 mesh or under is variously given at from 4d. to 9d. per ton. I take it that 60 % of the pulp can be reduced to 90 mesh for 5d. per ton, therefore it would cost 7½d. per ton to reduce the whole, but it might cost a 1s. per ton. Without tube milling, the sands residues were on the whole 4s. per ton, and by partial tube milling, reduced to something like 2s. 6d. per ton.

Before the introduction of the Usher process, the slimes residues would average on ordinary mines 1s. 9d. to 2s. per ton, so that there was little encouragement to alter the existing plant, but by the Usher plant the slimes residues of 1s. 9d. to 2s. could be reduced to 6d. per ton, hence a saving of something like 2s. per ton on sands becomes a possibility, even a certainty.

But that possibility is not to be achieved without considerable changes in plant. The Usher plant has very good points, but it has one important defect, it will not treat slimes laden with 75 % of sands, nor can it be regarded as a continuous process. I have, therefore, been gradually led to devise a plant to secure the double advantage of continuity and capacity to treat the product referred to—containing, say, 75 % fine sands and 25 % slimes. This plant consists of the ordinary decantation vats, provided with cone bottoms, and supported on cast iron columns, instead of a solid bed of mason work or concrete. The cost of construction would not be more than the former, if the excavations are taken into account, as no excavations are here required, and the outflow of slimes would be at least 6 ft. above the foundation line. Each vat is provided with an outflow at the bottom, and an overflow at the top. The entering sands and slimes are led into a distributor near the top of the vat, while the solution enters by a solution distributor placed about the middle of the vat. The vats are erected about 2 ft. above each other, rising from the first treatment vat to the last. The number of vats would depend on the richness of the ore, 4 vats in a row would deal with 2½ to 3½ dwt. pulp, 3 with poor slimes.

In operation, we suppose each vat charged with pulp which has settled in the cones, with solution above. The solution circulating from the top vat, say, No. 4, through each vat in succession to No. 1, whence it overflows to the boxes and after passing through, is continuously returned to No. 4. Operations would start by drawing pulp from the settling vat, by means of an injector, mixing it with the required amount of cyanide and distributing the pulp below the surface of No. 1 vat, here it meets with a slow uprising solution from the distributor, which is fed from the overflow of No. 2 vat. As it falls to the bottom it is robbed of some of its gold, and passing the distributor gradually descends in the cone, as the injector transferring to No. 2 vat withdraws the thick pulp at the bottom. The same process is then gone through in the other three vats, and in the final vat the thick pulp can be diluted with water, and washed to the dam. The peculiarity about the process is the settlement in the cone by which the pulp is greatly freed from water, and thus carries over the least possible amount of dissolved gold to the succeeding vat. The water consumption can be reduced to a minimum. I have had no opportunity of testing the process on a large scale, but the small experiments point to a successful issue. The plant would be arranged so as to vary the conditions, as required, and I expect to reduce the number of vats to three 30-ton vats, which would treat 150—200 tons per diem. The sands plant would be entirely done away with.

Notes on the Assay of Slimes from results by J. A. Jamieson, lately Assayer to the Adair-Usher Process Co.—The experimental work in connection with the introduction of the Adair-Usher Processes, necessitated the careful assaying of slimes, particularly residues. In spite of every care, differences did occur, and therefore a series of experiments was projected, to find out the cause of these discrepancies, and some of them were carried out. Unfortunately, the removal of the laboratory prevented the experiments from being completed. The results will, no doubt, interest assayers generally, and they are given here with due apologies for their incompleteness.

The questions to be solved were :—*A.* the loss in fusions, *B.* the cupellation loss, with special reference to slimes residues.

Perhaps some assayer may have an opportunity to check some of the following results and to supplement them. There is no doubt, that an important loss does occur in both operations, which, if it could be recovered, would be an acceptable addition to shareholders' dividends.

The estimation of the cupellation loss presents little difficulty, but the loss in fusions, as far as I am aware, has not been determined with any accuracy, but that such loss does occur, may be verified by re-fusing the slag with charcoal, etc. Possibly the following wet method might give satisfactory results with slimes. It has been successfully applied to rich ores, and to the analysis of cyanide solutions containing gold, yielding in both cases higher results than are obtainable by fusion. The process for ores is as follows:—

Weigh off 2 A.T. of the very finely powdered ore (if much pyrites is present roasting is necessary). Transfer to a 1,000 c.c. stoppered mixer, add 30 c.c. concentrated HCl and 5 c.c. bromine, allow to stand 1 hour, fill up to mark with water and add water to make up for volume occupied by ore $= \frac{\text{W in grammes}}{\text{specific gravity}}$. Shake up, allow to stand 15 minutes, shake up again, and filter through a folded filter (thick paper), take 750 c.c., boil off bromide, add Zn powder to precipitate the gold, allow to stand until all the zinc is dissolved, filter off the residue on to a small filter with the filter pump. The small filter should be prepared by previously passing through it 5 gm. finely powdered litharge suspended in water. After filtering off the gold, wash and aspirate until almost dry, fold the filter into small bulk, place on a small scorifier, surround with assay lead, and scorify to 5 gm., cupel and part as usual.

If the bromine is mostly absorbed in the first hour by excess of pyrites, 5 c.c. more should be added and allowed to stand another hour. Low results are obtained, if sufficient time is not given to allow the gold to be soaked out of the ore. If the pores of the small filter be not filled with PbO, finely divided gold passes through and even if the filter is good and PbO is not added, the results are liable to variation, as some of the gold surrounded by carbon (from the filter) escapes absorption by the lead.

The first four of the following experiments were made to show, firstly, if any difference occurred in using 2½ and 4 A.T. charges, and secondly to find out what the loss or gain would be in reducing small or large beads of lead. The fifth batch to find out the advantage or otherwise of quick fusion. A very rich sample of slimes was used in the expectation that differences would be more obvious than on ordinary slimes.

The assays were carried out as follows:—

The whole of the charge was weighed out and transferred to an enamelled can (the popular miners' tea can), provided with a close fitting wood lid, lined with rubber. Half a dozen rapid twirling shakes of the can suffices to obtain a uniform mixture of the contents without loss. New pots only were used, and these were damped inside and rinsed out with borax. Each batch was started in the cold and slowly raised to incipient fusion and allowed to remain in a pasty condition for at least 15 minutes The furnace was then raised to full heat, and the pots allowed to remain until the contents were perfectly quiescent and transparent. The cupellations were made without previous scorification in morganite cupels, sufficient silver being added to ensure clean parting, but without breaking up. The partings were completed in 15 minutes with 30% acid free from haloids, the prills washed with distilled water, and annealed. All the fluxes were dried and in fine powder, and as far as possible, the conditions were kept alike with each batch. The fluxes used as also the lead buttons obtained are given in grammes.

BATCH I.

No.	Soda.	Borax.	PbO.	Carbon.	Ore.	Pb button	Dwts. per ton.
1	36	12	45	2·5	1 A.T.	27	
2	,,	,,	,,	,,	,,	28	8·60
3	,,	,,	,,	,,	,,	30	8·80
4	90	30	,,	,,	2½ A.T.	38	9·00
5	,,	,,	,,	,,	,,	34	8·88
6	,,	,,	,,	,,	,,	34	8·72
7	144	48	,,	,,	4 A.T.	40	8·70
8	,,	,,	,,	,,	,,	35	8·65
9	,,	,,	,,	,,	,,	35	8·85

Time of fusion, 75 minutes.

BATCH II.

1	36	12	45	2·5	1 A.T.	37	8·50
2	,,	,,	,,	,,	,,	39	8·80
3	,,	,,	,,	,,	,,	37	8·70
4	90	30	,,	,,	2½ A.T.	39	8·80
5	,,	,,	,,	,,	,,	39	8·80
6	,,	,,	,,	,,	,,	36	8·80
7	144	48	,,	,,	4 A.T.	27	8·62
8	,,	,,	,,	,,	,,	36	8·75
9	,,	,,	,,	,,	,,	34	8·75

Time of fusion, 65 minutes.

BATCH III.

1	36	12	88	4·5	1 A.T.	71	8·30
2	,,	,,	,,	,,	,,	72	8·30
3	,,	,,	,,	,,	,,	74	8·50
4	90	30	,,	,,	2½ A.T.	70	8·96
5	,,	,,	,,	,,	,,	67	9·04
6	,,	,,	,,	,,	,,	69	8·96
7	144	48	,,	,,	4 A.T.	62	9·00
8	,,	,,	,,	,,	,,	60	9·00
9	,,	,,	,,	,,	,,	63	8·90

Time of fusion, 65 minutes.

BATCH IV.

1	36	12	88	4·5	1 A.T.	79	8·70
2	,,	,,	,,	,,	,,	80	8·50
3	,,	,,	,,	,,	,,	79	8·80
4	90	30	,,	,,	2½ A.T.	76	9·00
5	,,	,,	,,	,,	,,	74	9·08
6	,,	,,	,,	,,	,,	70	8·80
7	144	48	,,	,,	4 A.T.	68	8·80
8	,,	,,	,,	,,	,,	67	8·67
9	,,	,,	,,	,,	,,	67	8·90

Time of fusion, 50 minutes.

BATCH V.

1	90	30	58	2·5	2½ A.T.	50	8·64
2	,,	,,	,,	,,	,,	46	8·76
3	,,	,,	,,	,,	,,	47	8·64
4	,,	,,	,,	,,	,,	boiled over	
5	144	48	88	4·5	,,	71	8·60
6	,,	,,	,,	,,	,,	boiled over	
7	,,	,,	,,	,,	,,	71	8·68
8	,,	,,	,,	,,	,,	72	8·96

Time of fusion, 25 minutes.

These results confirm the general experience that a long fusion with large leads gives the highest and, therefore, most correct results, also that 2½ assay ton charges are best. For very low residues, however, the 2½ A.T. leads should be scorified in pairs, otherwise the prills are difficult to weigh correctly. Further experiments would be required to show why the 2½ A.T. charges come out highest.

Cupellation Losses.—In the cupellation of bullion only a small amount of lead is used to a comparatively large amount of gold, and the cupellation loss is insignificant. But when the amount of gold is minute as compared with the lead as in assaying stones, then the cupellation loss affects the results sensibly.

The following tests were made in the attempt to determine the loss, but are incomplete for the reason given above.

Preparation of the Test Pieces.—It is not practicable to weigh off pure gold in the minute quantities required, and evaporation of gold solution on lead foil gave very unsatisfactory results. Some fine gold was therefore dropped into molten lead, brought to the melting point of gold, stirred and quickly poured on to a flat iron plate, and this operation of melting and pouring repeated (perhaps better results would have been obtained by further repetitions of this process to obtain a perfectly uniform alloy). Two such alloys were prepared, 5 gm. being used for each test.

ALLOY A.

		Nominal weight.	
1	5 gm. cupelled and parted	1·2	= ·06 mgm f. gold
2	,,	1·2	
3	,,	1·2	
4	,,	× 1·1	assumed mean
5	,,	1·2	1·2
6	,, + 25 gm. lead	1·2	
7	,,	1·1	
8	,,	1·1	
9	,,	1·0	
10	,,	1·1	1·1
11	,, + 65 gm. lead	× 1·2	
12	,,	1·0	
13	,,	1·0	
14	,,	× 0·8	1·0
15	5 gm.	1·0	
16	,, + 65 gm. lead	× 1·0	
17	,, scorif. to 10 gm. then	1·1	
18	,, cupelled and parted	1·2	
19	,,	1·2	1·16

ALLOY B.

1	5 gm. cupelled and parted	3·3	= ·17 mgm f. gold
2	,,	3·4	
3	,,	3·2	
4	,,	3·4	assumed mean
5	,,	3·4	3·4
6	,, + 25 gm. lead	3·3	
7	,,	3·3	
8	,,	3·3	
9	,,	3·4	
10	,,	3·2	3·3
11	,, + 65 gm. lead	3·2	
12	,,	3·1	
13	,,	3·2	
14	,,	3·1	
15	,,	× 3·3	3·15
16	,, + 65 gm. lead	3·3	
17	,, scorified to 10 gm. and	3·4	
18	,, then cupelled & parted	3·3	
19	,,	3·2	3·30

As before stated, these results are not quite satisfactory, but they suffice to show that there is a cupellation loss, which is by no means insignificant. Thus: assuming the figures as correct then the loss on 1 A.T. charges allowing ·02 for 5 gm. lead would be on 30 gm. lead buttons: ·12 dwt. per ton; and on 70 gm. lead buttons: ·24 dwt. per ton. On 2½ A.T. and 4 A.T. charges this loss would be divided by these figures respectively.

As shown in the pot results, the higher results are obtained by using larger amounts of lead. Hence, if the difference of the cupellation loss is added as a correction between an assay yielding 35 gm. lead and one giving 70 gm. lead it may quite easily amount to 6d. per ton or more on residues.

The President: I am sorry to say that Mr. Adair is not very well, otherwise he would have been here to read his paper himself. Years ago I know that quite a number of us, at any rate, were under the impression that the last word had been said in connection with the cyanidation of gold ores. We were all mistaken, and even nowadays new ideas are coming forward very rapidly indeed. This Adair-Usher process is a case in point, and I think both Mr. Adair and Mr. Usher are much to be congratulated upon their persistency, and the courage which they have shown in developing and pushing this process to a successful issue. Now, this particular process is being quite largely used on the Rand, and I do hope that those of you who are using it will come forward, and benefit not only our members, but everyone along the reef, and off the reef, by letting us have your views and experiences. In conclusion, I move a very hearty vote of thanks to Mr. Adair for his very valuable paper on the new process.

Mr. J. R. Williams: I fear I am something like the ancient mariner, in that I have been on the sea for a very long time. It is now 15 months since I have had an opportunity of visiting the mines of the Rand. At the start of the Usher process I took a very considerable interest in it. It gave me the idea from the original drawings, that it was a step in the right direction, and I have no hesitation in saying that it is a very valuable addition to our knowledge. I should like to ask one or two of the workers a question, and then to reserve my remarks till a later stage. My reason is this, that at the starting of the cyanide process, long before the slimes treatment was dreamt of, everyone had some idea of the use of an oxidising agent, and one gentleman rushed across to Pretoria and patented every possible agent. That is a patent that Mr. Crosse and myself have found very useful. We call it the Pharmacopœia patent, because it covers everything. Our own experience on a large scale was very disappointing. When making laboratory experiments, we found that we got results far better with oxidising agents than without them, but in no instance on the large scale did I find that any of these oxidising agents had any effect. Not even to the extent of 0·1% could I trace any advantage. I should like to ask those gentlemen, before making any criticisms on the paper, whether they have been able to get any advantage to extraction by the use of umber?

Mr. G. Melvill: I have had a great deal of experience with this particular process, and experiments were carried out on a large scale on the Crown Reef, under the supervision of the Adair-Usher Co. When I saw the results of their experiments on a small scale, I certainly thought them marvellous. They used umber all through their experiments and certainly got some very good results, but after I took over the running of the plant I stopped the umber, and my results were quite as good as theirs. So I came to the conclusion that it would be cheaper to work without the umber. Having about the worst accumulated slimes on the Rand to treat, I tried the umber on a small and large scale, but found I got better results by the ordinary circulation with pump compressed air. I, therefore, came to the conclusion that no benefit was derived by using it.

The President: Did you visit the Nigel, Mr. Melvill?

Mr. G. Melvill: No, I was unable to attend.

The President: I asked that question because I noticed down there they were using permanganate of potash in connection with the treatment of bulrushes. Just alongside the Chilian mill, which they were using to grind up these bulrushes, was a bowl of permanganate of potash, and I presume that they find the addition of permanganate useful as an oxidiser of organic matter.

Mr. G. Melvill: I may say that there is a large percentage of bulrushes in the accumulated slimes at the Crown Reef. I have tried permanganate, but it would take such a quantity to deal with the organic matter that, though the extraction would be improved, it would not pay.

The President: What were your values?

Mr. G. Melvill: About 3 dwt.

The President: I may say that the value down at the Nigel is very much greater, about 7 dwt., and it has been as high as 1 oz. In the old days they had 5 oz. ore in their mills on certain occasions, so this high value would allow them to use more permanganate than you could. With regard to this matter, I had better explain that the old accumulated slimes of the dam are overgrown with quite a thick growth of bulrushes, and you cannot get at the slimes without taking the bulrushes, and I assure you, to see the mess that comes up in those cocoa-pans is enough to frighten any ordinary cyanide manager. The process is quite simple. They tip it into a Chilian mill, at the bottom of which is a grating; a constant stream of water carries off the slimes as it is disintegrated by the rollers and passes it through a screen of $\frac{1}{8}$ in. or $\frac{1}{4}$ in. mesh. There are boys standing by these screens, and every now and again they pull them out and knock them clear of the disintegrated bulrush matter. It was in this Chilian mill that the permanganate of potash was being used.

CYANIDATION IN THE MALAY STATES.

By H. F. Lofts (Associate).

Perhaps some account of a plant which happens to be the only one in these States, and is also situated on the only working gold mine there, and has also the peculiarity of having shiftsmen who are coloured men, may be of some interest to your members. It is attached to the Raub Australian gold mine, situated at Raub Pahang F.M.S.

The reef may be described as a quartzitic, lenticular, interbedded vein, and contains scheelite, antimony, bismuth and arsenic in the form of arsenical pyrites. The current tailings assayed 1·12 dwt. and they were too low grade and too full of antimony to treat. Both weak and strong solutions were tried after the tailings were water washed, and in each case no result was obtainable. However, as shown below, when the concentrates were separated by Wilfley tables, they were treated as slimes and showed a 78% extraction. In fact, the greatest trouble had been experienced in obtaining a payable extraction from the higher grade sands, which had been allowed to accumulate. It was found that the chief cause of trouble was antimony, of which sufficient remained attached to small particles of quartz to make the sands and concentrates very refractory.

The higher grade tailings were successfully treated under the conditions mentioned below:—

Buddlings: Assays, 16·35 dwt.; extraction, 70·5%; residue, 4·81 dwt.; treatment, 4 days; cyanide used, 1·05 lb. per ton; number of solutions run on, 34. This sample consisted of finely crushed sandy concentrates. It showed no acidity or refractory material. The strongest solution used was ·07%, and the average considerably below this. The treatment consisted in running through weak washes, which produced 66%, and with second treatment this was increased to 70·5%. The last three washes carried no gold. It should be noted that solution values went up after double treatment was used.

Another sample of buddlings gave the following result:—

Assay value, 16·80 dwt.; tons, 1·71; tons of solution run through box, 82·6; lime used, 90 lb.; strength of KCy solutions, ·05, ·13, ·18%; sump, ·21% KCy; the precipitation was very good, sump often being nil on assay; final residue, 3·04 dwt.; extraction, 82%.

Results on Concentrates from Wilfley Tables.—These are obtained from the ore now being crushed by battery. The greater part of the sample would pass 120 mesh screen and is black in colour, consisting of a large proportion of heavy metallic particles and much iron. The original assayed 22·40 dwt. Treatment was carried on as under:—

Sept. 26, water wash (neutral), Sept. 27, second wash (neutral). Agitation carried on with 1% KCy solution showed the presence of sulphides with acetate of lead test. This was continued until Sept. 29, Sept. 30, aerating followed by solutions ·09% and ·06% KCy. Continued until Oct. 2, then second treatment and oxidation from 7 a.m. to 10.30 a.m. Solution then came off free from sulphides. The strong solution was then run off and a weak solution added.

Weak washing was carried on till Oct. 6, when two samples of slimes were taken and washed, the result being a residue of 8·17 dwt. It was found that by ordinary agitation 32·5% was extracted up till the 30th. Then the extraction stopped. A new system was started, which included aeration, oxidation and agitation, similar to that produced by a centrifugal pump. This brought the second residue down to 9·18 dwt. or 59% extraction, and finally to 8·17 dwt. or 63·5%. On a working scale, this extraction was brought up to 68%.

I now append details of another experiment on concentrates:—Assay value, 22·96 dwt.; length of treatment, 7·33 days. The product consisted of concentrates from the present mill pulp and contained a large quantity of antimony. The first two solutions carried no gold, showing ferrous hydrate or some other oxygen absorbing substance. Continuous washings with weak solutions and afterwards with ·35% KCy were very effective and gave an average of 2 dwt. of gold per ton until extraction stopped. A second treatment was resorted to, and the final extraction was as high as could reasonably be expected. The solutions came off clear, and the leaching was good. As there was no acidity no lime was added. I think caustic soda might with advantage be used to dissolve organic matter and avoid a heavy loss of cyanide.

This experiment was a sands treatment on 100 lb. of ore. Final residue, 1·14 dwt. = 95% extraction. The extraction obtained from 372 tons of concentrates and buddlings treated equalled 77·1%, average number of days, 7·77; average charge, 9·41 dwt. The highest extraction being 93% and the lowest 69·8%. These results were obtained from a small plant with a capacity for slimes equal to 200 tons per month. The above plant, through anything but large, is sufficient to show what results could be obtained with a plant of greater capacity.

Precipitation.—The solutions in use on concentrates which assayed from 10 dwt. to 38 dwt. and carried from ·18 to ·05% KCy after passing the boxes varied from a few grains to nil. With lower grade slimes, which contained free and fine gold, solutions were used down to ·001% KCy, which gave complete precipitation in the zinc boxes. In these cases the zinc was coated with acetate of lead. In dealing with stronger solutions no acetate of lead was necessary, as good extractions were obtained. The acetate of lead had the effect of making the bullion very impure. Sodium cyanide was used, the average consumption being 1·60 lb. per ton. Large quantities of a white precipitate occasionally formed in the boxes. When this occurred it was found to be necessary to increase the strength of the solutions by adding solid cyanide at the head of each box. This dispersed the precipitate, which only formed when a weak solution was used. The cyanide solution became at times very foul, and experiments showed that it was advisable to increase the alkalinity by adding caustic soda to the sump and solid cyanide at the head of the box. To ensure a good precipitation an average of 80 lb. of lime was placed in the vat with each charge of 10 to 12 tons. With solutions of the strength mentioned in the preceding paragraph, the boxes were kept fairly free from white precipitate.

Antimony.—In this connection it is interesting to notice how quickly when powdered, this is acted upon by caustic soda. The gold slimes were of low grade in the extractor. This, I think, is the usual experience, when ores of this nature are dealt with owing to the precipitation of base metal with the gold. Owing to the minute quantities of almost invisible slimes which were not settled with the lime and which coated the zinc and stopped precipitation, it became necessary to introduce sand filters, which were very effective

Agitation.—It was found that a long agitation was not an advantage, a short agitation lasting one hour producing a solution as rich in gold as one which lasted five hours. The usual proportion of slimes to solution was 1 to 3.

Tonnage Estimates.—I found that, if after decantation I measured the cubic feet of wet slimes in the vat and divided by 30 (= cubic feet per ton of slimes) and deducted 33% of moisture, it gave an excellent agreement with samples taken when the vat was full and in agitation.

The President: I move a vote of thanks to Mr. Lofts for his paper, and in doing so, I would say that it always gives us special pleasure to receive papers from members oversea.

NOTES ON TIN MINING IN CAPE COLONY.

(Read at December Meeting, 1907.)

By Harry D. Griffiths, A.R.S.M., Wh.Sc. (Member).

DISCUSSION.

Mr. J. R. Williams: I would like to make a remark on this paper. I have heard it repeatedly stated, especially lately, that the assayers of the Witwatersrand, in fact in the whole of South Africa, are very indifferent about tin, and that a lot of them fail to find tin, when there is plenty. I remember a case when a sample of so-called tin was brought to me. The man was a Cornishman, at least, I think so, and he told me that the sample contained 70% tin, and that he had two certificates to the effect that there was no tin in it. I thought that a Cornishman should know tin when he saw it, and so I asked my assistant, who told me there was no tin present. I thought I would try it myself, and I applied a method which, I think, never fails to give you tin, when present in the ore. I have spoken lately to one or two assayers concerning this method, and strange to say, it had escaped their memory. What this Cornishman took to be tin was undoubtedly a form of titanic iron. When crushed and on the shovel it certainly looked very much like tin.

The method is simply one of reducing to metal, by ignition with gas or hydrogen. Then it is very easy to dissolve it with hydrochloric acid. Many years ago I was asked to examine this method, and I did so by adding tin stone of a known value to all the samples. It always gave very accurate results. Mr Caldecott informs me that Beringer gives the method, and I think it will prove a good check on any method they may use for these obstinate samples.

Mr. J. Gray: With reference to this method of assaying, I have tried it and find that the essential part consists of getting the ore into a state of very fine division. Parry's book deals with this matter very fully, and he recognises it as the best method, though there is some doubt about it.

Mr. A. Whitby: This method has always been looked upon as the final court of appeal. It has been supposed to be the most perfect method of tin assaying. I have found hydrogen more effective than local coal gas in reducing the tin. After reduction and solution in hydrochloric acid the process branches in several directions, some operators precipitating the sulphide and weighing the oxide from it; others titrating with iodine, and others again reducing with zinc to a sponge of metallic tin, oxidising with nitric acid, and weighing the resultant oxide.

THE ORIGIN OF THE GOLD IN BANKET.

(Having reference to Prof. J. W. Gregory's paper on the same subject, read before the Institution of Mining and Metallurgy.)

(Read at January Meeting, 1908.)

By J. S. Curtis (Member).

REPLY TO DISCUSSION.

Mr. J. S. Curtis: In regard to Mr. Milligan's observations of February 15, I am quite of his opinion that the quartz veins could have no influence on the deposition of the gold.

I have to thank Mr. T. Lane Carter for his flattering criticism of my paper, and I agree with him in what he says about Prof. Posepny of Bohemia. Still, able as he is, he can scarcely be called "the father of the modern science of ore deposits," as that title undoubtedly belongs to Bernhard von Cotta (Professor of Geology in Freiberg), who, in the year 1853, published *Die Lehre von den Erzlagerstätten.* I had the pleasure of attending his lectures at Freiberg in 1867 and 1868, and can bear witness to the great reputation he had as a mining geologist. I believe he was the first writer, who combined the science of geology with a practical knowledge of mining.

I can quite understand that Prof. Posepny, from the limited information he had at hand when he published his "Ore Deposits," formed the opinion that the gold of the Rand was placer gold. I had also formed the same opinion in Barberton before I came to Johannesburg in 1887, but soon had reason to change it. In 1886 I had the pleasure of corresponding with Prof. Posepny in reference to my work on the "Silver Lead Deposits of Eureka, Nevada" (VII. Monograph of the U.S. Geological Survey), and regret very much that after I came to South Africa I was unable, through want of time, to keep it up.

As regards "the bigger the pebble the richer the ore," what I wished to convey was that, as a rule, a gold bearing banket reef was richest in those places where the pebbles were largest, not that a reef carrying large pebbles should necessarily be a good gold producer.

I quite appreciate Dr. Moir's remarks on the instability of gold compounds, and also what he says about the "critical temperature" of water, but I think that pressure would have a very marked influence in keeping the numerous compounds in solution, even at a temperature above 330° C. The rock at the time the depositions took place could be compared to a leaky boiler in which it might nevertheless be possible to keep up a very considerable pressure.

ASSAY OF BATTERY CHIPS AND SCREENS.

(*Read at February Meeting, 1908.*)

By Leslie J. Wilmoth (Associate).

DISCUSSION.

Mr. H. R. Jolly : In offering a few remarks on this paper, it is not my intention to theorise, but to give a few notes embodying the practical results of some experiments carried out at the Robinson Deep on this class of material.

The samples of battery chips received, were divided into three grades :—

1. Fines − 20 mesh.
2. Intermediates + 20 − 10 mesh.
3. Metallics + 10

The last measured in some cases about 1 in. in diameter, and were only slightly coated with a film of oxide, which could be easily removed by rubbing between the fingers. I may state that we have tried the author's method of assay, and find it to be very unsatisfactory indeed, in spite of the fact that we followed his directions, and were careful to wash the slags down with litharge and carbon. It is only fair to state that his slags were fluid enough, but the real difficulty presented itself when we tried to detach the lead buttons from the matte. In many instances the lead was so brittle, that it broke away from the matte in pieces, and the cupels showed considerable quantities of iron oxide after cupellation, leading to the conclusion, that there must have been a large quantity of iron or iron sulphide carried down with the lead. The mattes, from six assays of our fines, were then run down together and were found to contain 72·0 dwt., equal to a loss of 12·0 dwt. of gold per assay. The slags only contained sufficient gold to account for a loss of ·20 dwt. per assay. Three other assays by this method showed losses in the mattes only equal to 1·0 dwt. per assay, but the slag loss amounted to 6·0 dwt. per assay. In these cases, however, the mattes formed were very small.

While treating a sample some 12 months ago we discovered that sodium bisulphate was very useful in fluxing iron, and conducted a number of experiments with it, the following fluxes being the nett results of our investigations :—

	1 Fines.	2 Intermediates.	3 Metallics.
Charge	1 A.T.	1 A.T.	1 A.T.
Bisulphate	¼ ,,	½ ,,	¾ ,,
Soda	2 ,,	1 ,,	1½ ,,
Silica	1½ ,,	1½ ,,	1½ ,,
Borax	1 ,,	1 ,,	1 ,,
Litharge	1¾ ,,	1½ ,,	2 ,,
Nitre	1 gm.	2 gm.	4 gm.
Wash	½ A.T. PbO carbon	½ A.T. PbO	½ A.T. PbO
Wgt. of Pb.	50 gm.	40 gm.	60 gm.
Average A.V	26·0 oz.	23·0 dwt.	18·00 dwt.

The various charges were run down in H pots, well packed round with coke, the furnace being urged to its fullest extent. Immediately the fusions settled down, the washes, as stated, were added, and the pots kept at a high temperature for other 20 minutes. When the furnace is going well, one hour is found to be sufficient for the fusion. Bubbles of gas come up before the wash is added, but disappear afterwards. The slags were all very fluid, clean and glassy, and in all cases the lead buttons were soft and malleable, and broke away from the slags nice and clean. The cupels showed no signs of iron having been present.

The reason for only using litharge as a wash with the intermediates and metallics is that, in these cases, there is generally sufficient sulphur reduced from the bisulphate to act upon the litharge. At the same time, we have found it advisable not to use too much nitre, as an excess does not reduce the size of the buttons but merely acts corrosively. Some nitre, however, is essential in preventing the reduction of the bisulphate to sulphide, which tends to pass sulphur into the buttons. Our slags from the fines were found to contain gold equivalent to an average loss of 2·0 dwt. per assay, which appears to be a very decided improvement on the matte method of assay. As a severe test of our method we ran two fusions, one of which consisted of six pieces of iron, and the other of eight, to the assay ton and found no difficulty in getting good fusions with homogeneous slags.

It is advisable to put on a trial assay, as slight alteration in the fluxes may be necessary for different samples, mainly in the litharge and soda. The litharge we use is in a higher state of oxidation than the ordinary yellow kind, and is really a mixture containing red lead. The formation of a matte at any time in gold assaying is, in my opinion, bad practice and should be guarded against. In fact, I do not quite see how it is possible to form a matte, which will be gold free. The simpler the products of a fusion are the better.

With reference to the assay of battery screens, I quite agree with the author's preparation of the sample by oxidation and crushing of the oxide formed, as more concordant results will be obtained by that method than by assaying direct. I disagree with him, however, in his practice of forming a matte in the assay of the oxide. The fluxing of the oxide is a very simple matter indeed, and one which can be treated almost like an ordinary assay. As far as I am aware, sodium bisulphate has never been put to any use in the assay of gold before, and I think that there is a possibility of its finding a more extended application as a flux.

In conclusion, I wish to draw attention to the fact that there are small quantities of the metals of the platinum group present in the samples of battery chips, as also in black sands.

Mr. H. A. White: There is one point in connection with Mr. Jolly's contribution to this discussion, that I should like to draw your attention to. The peculiar flux recommended contains bisulphate of sodium and sodium carbonate in the same charge. This cannot be defended. If the intention is to add sodium sulphate (neutral), this round about method is not scientific, for the carbon dioxide evolved can be of no service and merely increases the chances of loss through boiling over or spitting. The appended equation will render my meaning clear

$$2NaHSO_4 + Na_2CO_3 = 2Na_2SO_4 + CO_2 + H_2O.$$

I do not undertake to say that Na_2SO_4, which apparently remains unaltered during the fusion, will have any useful effect in such assays as are quoted.

Mr. A. Whitby: It was not my intention to contribute anything to this discussion, for the reason that not having had any opportunity to carry out experiments with the author's method of assaying, I did not feel justified in actively criticising it. But there is a somewhat interesting point brought out in Mr. Jolly's contribution, which may be worthy of remark. It may be said that the author's and Mr. Jolly's methods represent the two extremes of our conception of fluxing, the former removing the iron as a sulphide and the latter as a sulphate. Although the first does not say how much litharge he uses, it is evident from his remarks that he adds a quantity merely sufficient for his button and, in the added sulphur, the iron of his charge, the nail and the charcoal, has a great preponderance of reducing agents. The second uses no charcoal except in his wash for fines, and, with the sulphuric acid of the bisulphate the added nitre and the oxygen of his litharge, appears to have an excess of oxidising agent, merely relying on the iron of his charge for the production of a button of lead. These methods are so diametrically opposite in character that one finds difficulty in conceiving, that they should arrive at similar conclusions. Many years ago in this Society,* one assayer recommended a flux of carbonate of potash, litharge, borax, salt, excess of reducing agent, and an excess of iron in the fluxing of pyritic ores as a means of preventing the carrying of silver and gold values into the slag. This was on the assumption, that a ferrous potassium sulphide was formed, but I have not found any further reference to this conception of fluxing, and my own opinion about the assay of moderately pyritic ores is quite antagonistic to such a method. And this inclines me to think that in the generality of cases the author's process must fail, for the following reasons.

I wish to definitely state here the position taken up by me with regard to this matter. Whether it is the carrying of iron or of sulphur into the slag, any condition which disposes the slag to retain alkaline sulphides seems to me to imply the retention of gold, since the metals of the platinum group share, in some measure, the property possessed by those of the tin group of forming sulpho salts. By observation and analogy, it is rare to find a matte entirely free from the precious metals. It is difficult to so proportion the reducing agents, that each shall fulfil its own particular duty without interfering with the others, an excess of sulphur, for example, implying lead sulphide in the button whilst the iron nail, according to my experience, when working at high temperatures, has an inconvenient habit of melting into the lead button with disastrous results on cupellation. Further, it is the exception rather than the rule to find a well defined line of demarcation between the button and the matte.

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* See this Society's Proceedings, vol. ii., p. 601.

Mr. Jolly's method seems to offer a more reasonable solution of the difficulty presented by this class of material, and is quite in accord with the principle of assaying ores containing pyrites, where the sulphur, in my opinion, is best oxidised to the sulphate state. A little comparison will show, that in this case also, practically all the lead is reduced.

In reference to Mr. White's remark, one would certainly premise, that he is correct in his statement, but the facts appear to be that Mr. Jolly has tried the neutral sulphate without the beneficial effect of the acid salt. In making acid sulphate fusions of iron ores, although this is scarcely parallel, I have observed, that solution of the iron only takes place at a temperature when presumably all the extra sulphuric acid has been driven off, whilst even with a full red heat the fumes of SO_3 are still noticeable. It might, therefore, be assumed, with some little justification, that the acid sulphate scarely enters the field until the alkaline carbonate has been, to some extent, satisfied by earlier reactions.

From this point of view I cannot agree with the author's treatment of his roasted product from battery screens, and should certainly prefer to carry the iron into the slag, as silicate and borate, to going out of my way to form a matte with it. With particular reference to the matte assaying of cupriferous gold-bearing ores, I would call the author's attention to Peters' observations on this point in his principles of copper smelting. As there expressed, it is the opinion of the great majority of metallurgists that the existence of very small amounts of copper in the ore is sufficient to carry the gold values into the matte in matte smelting, since copper possesses a remarkable affinity for gold. I do not here mean to imply that the author did not recover the whole of his gold in his experiment with copper ore, but merely to point to the possibility of failure under dissimilar conditions.

WESTRALIAN WET-CRUSHING PLANTS, WITH SOME NOTES ON LABOUR EFFICIENCY.

(*Read at February Meeting, 1908.*)

By Gerard W. Williams, F.C.S., A.I.M.M.

DISCUSSION.

Mr. F. F. Alexander: The author's description of the general practice obtaining in Westralian wet crushing plants is certainly most interesting to most of us, who knew him here, but I certainly consider him unkind, to say the least, in recommending for our notice such time honoured abuses as inside amalgamation, grinding pans, and 3 ft. 3 in. tube mills, to say nothing of the apparent undermanning of the plant at the expense of a reasonably good extraction. I believe inside amalgamation is still practised on some of our out of the way plants, and the advantages claimed for it is a myth. In other words, by neglecting adequate plate dressing for the constant feed of mercury and contaminated gold from the mortar boxes you are losing the substance for the shadow through faulty amalgamation. I cannot agree, that a tube mill is essentially an impact grinder. That impact assists within small limits is a reasonable assumption, but I certainly class the tube mill with attrition grinders, I would go further, however, by classing a 3 ft. 3 in. × 13 ft. mill as a worthless installation—the load in so small a mill, not being heavy enough to give full crushing capacity. Neither is the inclined plane or race of pebbles long enough to do the work, and I feel sure a 6 ft. × 18 ft. 6 in. mill with peripheral discharge would yield results which would astonish the advocates of grinding pans. I will not attempt to defend a possible misuse of unskilled labourers on these fields, but I certainly believe in a judicious admixture of such labour, as we have at our disposal. A numerically increase of the total number employed on Westralian plants would tend both to reduce costs and improve extraction It is a well-known fact that our Kafirs are quite useless if left to their own initiative on work that is not absolutely routine, and knowing this, the skilled workman is saddled with the responsibility of his own work plus that done by a most inefficient gang of irresponsible Kafirs. These facts have to be taken into account, and sorely trouble foremen and shiftmen alike on a large reduction plant. Is it not a penny-wise policy to expect your assayer to light fires, fetch and carry, when more important duties have to suffer? The same might be said of cleaning and sweeping by the extractor house man to the neglect of proper classification of solutions, etc., affecting cyanide consumption and extraction generally. In conclusion, I am pleased to note the author allows, that our costs are low and extraction high, and it is most gratifying to know that these results are obtained with labour far below the standard of Westralian efficiency, according to his showing.

Mr. E. M. Weston: Regarding the author's remarks on the number of men employed firing boilers on the Ida H. G. M. Co., he says one man is sufficient to fire four boilers, but he omits to state that these boilers are fired by wood. There are few ashes to remove compared with the 30% here, and a man can fire with wood much more easily than with coal. Comparisons are useless, when all the facts relating to them are not stated.

NOTES ON RAND MINING.

(Read at March Meeting, 1908.)

By Tom Johnson (Member).

DISCUSSION.

The President: Mr. Richardson will kindly read a contribution by Mr. Phillips on Mr. Johnson's paper. I have here a letter from Mr. Phillips, stating that this is only the first instalment.

Mr. J. M. Phillips: As our President has suggested, the author has certainly introduced nearly all the contentious points of mining in his paper, and on each one a good "paper" could be written. He covers such a large field, and generalises so much that one hardly knows where to begin in discussing the paper. In the first place, he contends that there are too many shafts and winding engines, etc., for the amount of rock hoisted. I agree with him, that there have been more shafts sunk than are absolutely necessary at the present time, but when the greater majority of those shafts were sunk, or in process of sinking, the true value or dip of our reefs were not so accurately known, as at present. Certainly no one can accuse us of sinking too many shafts on the one property at the present time. In fact, I contend that we are not sinking enough for proper ventilation and development. The idea at present appears to be, that one large shaft is more economical than two smaller ones, but I fail to see where the true economy comes in. In the one shaft system all the tramming, hoisting and ventilation is concentrated on one point. There is generally a congestion of traffic underground in consequence, and in the event of a serious accident happening to any one of the compartments, in all probability it will affect the whole shaft, disorganising both hoisting and milling. All the necessary engines, three or four, are located under the same roof, which may easily result in the confusion of shaft signals, especially when two or more are being given at the same time. In my opinion, the adoption of two smaller shafts is more economical and advantageous, for if started simultaneously the development is better and more speedily accomplished, more faces, as the author says, being attacked at one time and ventilation more quickly secured.

The two shafts should not be so far apart as at present, as in the event of an accident occurring to the one shaft, the second one would be easily accessible for all purposes. Whilst I am in favour of the two smaller shafts as against the one large one, yet I do not believe, as the author evidently does, that three-compartment shafts are large enough. His contention that 5,000 tons can be hoisted through two 6 x 5 ft. compartments in 24 hours is feasible enough for, say, one 24 hours, but I doubt very much if the rate can be kept up day after day the whole month through, or even for one week. I work it out at approximately, with 4-ton skips, 200 tons per hour, or 25 trips with each skip, which means a trip every 4 minutes; this, I own, is not an excessive speed for any one trip, but when it comes to having to maintain this average the whole month through I think you will find that it will tax the energies of the skip man, at least, to his full capacity. Further, there is no allowance made at this rate for unavoidable loss of time, such as is needed in changing from station to station, examining the engines, shafts and ropes, etc. Then again, presuming that this rate can be maintained, how, may I ask, are the men, tools, etc., to be raised, which are necessary to break the 5,000 tons; surely, not by the same engine and through the same compartments, for I take it for granted that the third compartment is reserved for pumps, ladder way, etc. I have been on mines here, where it takes a separate engine and compartment for this work alone. In my opinion, nothing smaller than a four-compartment shaft will meet the requirements; two compartments for hoisting rock, one compartment for men, tools, etc., one compartment for pumps, ladder way, etc., and, of course, for the men, tools, etc., an extra engine is necessary.

As regards the shape of the shaft, I am in favour of the rectangular as against the circular. True circular shafts are stronger and safer in weak ground, but on the Rand—with few exceptions—we have no "heavy" ground to contend with in sinking, and when the shaft is sunk across the formation it is seldom necessary to carry "full" sets the whole depth. The hoisting in rectangular shafts certainly retards the ventilation to some extent, but not to the same amount

RULES

RELATING TO THE

TRANSVAAL
STOPE DRILL COMPETITION,

1909.

PRELIMINARY.

The Transvaal Stope Drill Competition is being organised by the Transvaal Government and the Transvaal Chamber of Mines, which are jointly contributing the prizes, and have nominated a Joint Committee which has drawn up the conditions under which the Competition is to be held, and which will arrange and carry out the Competition. A list of the members of the Committee will be found on the last page.

The object of the Competition is to obtain a small drill capable of economic use in the narrow stopes on the Witwatersrand which are at present worked for the most part by hand labour. With this object in view, the Committee has endeavoured to make the conditions as practical as possible.

To reduce the number of competitors within manageable limits, power is taken to reject or eliminate entries which are obviously outclassed or unsuited to the existing mining conditions.

In the final competition, mining conditions must be taken to mean work in stopes varying in width from 20 inches to 48 inches, and with a dip varying from 20° to 90°. It may be assumed, further, that probably 90 per cent. of the holes drilled will be down holes.

The Committee anticipate that all or most of the machines entered will be worked by means of compressed air, the most usual method on the Witwatersrand; but, should any entries offer of machines worked by other kinds of power, the Committee will consider such machines.

Competitors who desire to protect their patent rights must make their own arrangements for obtaining provisional protection.

Johannesburg, May, 1908.

CONDITIONS OF THE TRANSVAAL STOPE DRILL COMPETITION.

A.—GENERAL.

Interpretation.—1. In these conditions the following expressions are used, with the meanings set against them:—

"The Committee" means the Technical Committee appointed jointly by the Transvaal Government and the Transvaal Chamber of Mines to arrange and carry out the Competition, or any duly appointed Sub-Committee or representatives of the Committee.

"Competitor" means the person, firm or company entering a machine for the Competition. A competitor will be treated as a separate competitor in respect of each type of machine entered by him, the Committee deciding what constitutes a different type.

"Machine" means the whole drilling machine, without supports or dust-allayer, provided the supports and dust-allayer are not integral parts of the machine, and does not include the drill.

"Drill" means the jumper.

"Bit" means the cutting end of the drill.

"Elimination trials" means the preliminary trials on surface and underground.

"Competition" means the final trial underground.

Committee's Decision Final.—2. The decision of the Committee in all matters relating to the trials and the competition shall be final. Competitors shall be bound by these conditions. The Committee reserves the right to relax, alter or add to the requirements of any of these conditions should it think it desirable in any particular case. The Committee reserves the right to repeat the trials or competition or any part thereof in respect of any entry should it think fit.

Disqualification.—3. The Committee may disqualify any competitor in respect of any or all of his entries, if satisfied on enquiry that he has interfered in the conduct of any of the trials or competitions or has corrupted any miner, or has attempted to do so, or if he refuses to comply with any conditions laid down by the Committee. No prize shall be awarded to any entry in respect of which any competitor is so disqualified.

B.—ENTRIES.

Date of Entries.—4. Entries must be received on or before December 31, 1908, by the

"Secretary to the Committee,
Transvaal Chamber of Mines,
Johannesburg,
Transvaal."

Entries must be made on a form which may be obtained at the above address, or from the

"London Secretary,
Transvaal Chamber of Mines,
202, Salisbury House,
Finsbury Circus,
London, E.C."

Specifications and Price List.—5. All entries must be accompanied by:—

(a) Detailed specifications and drawings.

(b) A list of prices of the machine and accessories, including drills, bars and arms (where such are required) and spare parts, and an illustrated and numbered list of spare parts in triplicate.

(c) A guarantee to the satisfaction of the Committee to supply machines and accessories in Johannesburg at the price quoted in the price list, subject to ordinary trade fluctuations in the price of materials.

(d) With respect to the conditions contained in paragraphs (b) and (c) of this clause, if an intending competitor is unable to assign prices to the machine and its accessories, he shall at the time of his entry notify the Committee in writing of this fact, and also of his acceptance of the following conditions:—

(1) The machine and accessories shall, for the purpose of the Competition, be valued by the Committee.

(2) The competitor shall enter into an agreement to allow the Transvaal Chamber of Mines or its nominees to manufacture and use the machine and accessories at a royalty of a percentage on actual cost, such percentage to be stated by the competitor at the time of entry.

Representation of Competitors.—6. Each competitor shall at the time of entry nominate a representative for all purposes of the trials and competition, who may be either himself or another person. Such representative must be in Johannesburg throughout the trials and competition, and may be present with his machine at any time. The Committee may allow the appointment of an alternate.

Weight of Machines.—7. No entry will be accepted in respect of a machine weighing more than 100 lbs. It is desirable that the machine should be lighter than 100 lbs., as the need of the industry is for a one-man machine.

Delivery to Committee.—8. All machines, accessories and parts intended for use in the trials or competition (other than drills) shall be personally delivered free of cost in Johannesburg by the competitor's representative to the Committee not later than January 31st, 1909, at such place as the Committee shall appoint, and no machines, accessories or parts not so delivered shall be used in the trials or competition, except as desired by the Committee.

Care of Machines.—9. All machines, accessories and parts so delivered shall remain in the possession of the Committee, which shall not be responsible for loss or damage in the course of the trials or competition, and which reserves the right to retain any machines or spare parts for the purpose of testing materials.

Box for Machine.—10. Each machine shall be supplied in a strong box fitted with a padlock, and the accessories and spares shall be supplied in a separate box with padlock.

Dust Allayers.—11. Competitors must supply with each machine a suitable dust allayer, which may be either a part of the machine, or attached to it, or separate.

Number of Machines.—12. Each competitor must supply three machines to the Committee, and may supply such other machines and spare parts as he may desire, which shall be kept in the custody of the Committee and issued for use only with the Committee's consent.

Drills.—13. A Competitor may supply his own drills, and may change the supply during the competition on complying with the conditions of Clause 5, but the Committee reserves the right to decline to use any drills so supplied if it considers them unsatisfactory.

Irregularities.—14. No entries will be accepted which do not fulfil the conditions of Clauses 4 to 13 inclusive.

Rejection of Entries.—15. The Committee reserves the right to reject the entry of any machine which appears to it, from the drawings and specifications, to be unsuitable in material and design.

Withdrawals.—16. No entries may be withdrawn except by permission of the Committee.

C.—ELIMINATION TRIALS.

Object of Trials.—17. To reduce the entries for the six months' underground competition to a number that can be conveniently handled, elimination trials will be held successively on the surface and underground.

Conditions of Trials.—18. The following conditions will apply to the elimination trials, both on surface and underground :—

(a) The competitor may, if he so desires, provide his own operator, but the Committee reserves the right to put the machine to such tests under its own operators as it may think fit.

(b) All holes drilled shall not be less than 42 nor more than 48 inches in depth (no credit being given for any depth beyond the maximum), and shall be drilled with not more than four steps of gauge. The size of the starting and following bits will be left to the choice of the competitor, but the last bit used shall gauge at least 15/16 inch at the conclusion of drilling.

Surface Elimination Trial.—19. The following conditions will apply to the surface elimination trial only :—

(a) Two machines of each entry, or more at the discretion of the Committee, will be tested as to their efficiency in drilling down holes in granite blocks.

(b) Competitors may decide the air pressure to be supplied, which shall not be less than 60 lbs. nor more than 75 lbs. per square inch. In the absence of any requisition from the competitor, the pressure will be 75 lbs. per square inch.

(c) Each machine will be tested for one hour actual drilling time, provided that a total of two hours for rigging up, changing drills, drilling, etc., is not exceeded.

(d) The air consumption of each machine will be measured, and will be taken into account in deciding whether such machine has passed the elimination trial.

(e) All entries will be eliminated which cannot drill a minimum of one inch per minute of actual drilling time.

Underground Elimination Trial.—20. The following conditions will apply to the underground elimination trial :—

(a) Only machines which have passed the surface elimination trial will be allowed to enter.

(b) One machine of each entry will be subjected to this trial.

(c) The trial will last three periods of eight hours each, such periods to include time spent in changing drills, rigging up, etc.

(d) All trials will, if practicable, be conducted in the same stope at the air pressure prevailing, which shall be not less than 60 lbs. nor more than 75 lbs. per square inch.

(d) A machine will only be allowed to drill three holes per rig-up, each rig-up to be at least 15 feet from the previous one.

D.—THE COMPETITION.

Entries.—21. At the end of the underground elimination trial, the Committee will arrange the entries taking part therein in order of merit with regard to footage drilled and air consumed, and will then decide which may enter for the competition.

Date of Competition.—22. All machines not eliminated in the previous trials will be entered for a competition under practical working conditions underground, which will begin about April 1st, 1909, and is expected to last six months.

Duration.—23. Two machines of each entry will be run for 300 shifts each, by day and night, continuously, with the exception of Sundays and legal holidays. The length of shift will be of equal duration on each mine.

Lost Time.—24. If any entry of which two machines, in spite of the provisions of Clauses 32 and 35, fail to complete 300 shifts each, the footage drilled shall be charged with the estimated cost of 300 shifts; but time lost through no fault of the machine may be made up with the consent of the Committee.

Choice of Mines.—25. The mines and working places in which the competition will be carried out shall be selected by the Committee. So far as practicable, each entry will be run for an equal period in each stope in each mine selected.

Method of Operation.—26. Each pair of machines will in each shift be in the charge of one white miner selected by the Committee, with such native assistance as in the opinion of the Committee is required.

Control of Work.—27. During the competition the machines will do ordinary mining work, in accordance with instructions given to the miners from time to time by the Committee. Not more than two holes on any one rig-up shall be counted, except with the express sanction of the Committee. Any number of bars of any length may be used. As far as practicable the conditions under which the different machines work shall be kept uniform.

Pillars.—28. The competing machines will not be required to cut pillars, box holes, etc.; this will be done by the mines.

Control of Spares.—29. The miner will be forbidden to use any machine or spare parts except those issued to him by a representative of the Committee.

Issue of Spares.—30. A reasonable supply of spare parts will be issued to each miner from time to time by the Committee as required for each machine operated by him. The miner will sign a receipt for all such issues, and will keep them under lock and key in a box provided by the Committee.

Return of Worn Parts.—31. All worn and replaced parts shall be returned by the miner to the Committee not later than the end of the shift in which they were thrown out. A label, signed by the Committee's representative, shall be attached to each such part, showing the period for which it has been in use.

Issue of Machines.—32. If either of the pair of machines at work is so disabled as in the Committee's opinion to require to be sent to the surface for repair, a third machine, supplied by the competitor and stored in the working place, shall be issued by the Committee to take its place, but such third machine shall be returned as soon as the repair is completed. Machines supplied by the competitor in addition to the three required by Clause 12 shall only be issued with the special sanction of the Committee.

Removal of Machines.—33. When any machine is to be moved from a mine or working place in which it has been tested to another, it shall be handed over, together with all spare parts in hand, by the miner to the Committee, and checked by the latter. The removal will be entirely under the control of the Committee.

Air Pressure.—34. No machine will be operated at a pressure exceeding 75 lbs. or less than 60 lbs. per square inch at the end of the pipe line, as shown by recording gauges.

Air Consumption Tests.—35. The air consumption of each machine will be tested on the surface at least once a month during the competition, at intervals chosen by the Committee to afford a good knowledge of the average consumption by the machine underground. While a machine is withdrawn from work for this purpose, its place will be taken by a third machine supplied by the competitor. Should this for any reason be impossible, the time lost may be made up.

E.—BASIS OF AWARD FOR MACHINES.

First Prize.—36. The first prize of £4,000 will be awarded to that entry for which the total cost when divided by the total footage drilled by all machines of that entry during the competition is least, the Committee reserving the right, in case of two or more entries showing approximately equal results, to ignore small differences in points and award the prize to that entry which it considers best suited for the Rand, having regard to weight, general design, ability to drill deep holes, freedom from stoppages and breakdowns, facility of handling, and suitability to narrow stoping.

Second Prize.—37. The second prize of £1,000 will be awarded to that entry which is considered by the Committee, on the basis laid down in the preceding paragraph to be second best.

Measurement.—38. The footage drilled shall be measured at the end of each shift. No hole of a depth of less than 42 inches shall be counted, except where authorised in special circumstances by the Committee's representative, provided that any hole fitchered at a depth of not less than 30 inches shall be counted. The last bit used shall gauge at least 15/16 inch at the conclusion of drilling.

Items of Cost.—39. For the purposes of Clause 36 "Cost" shall mean the cost of:—

(a) First cost of machines and rig-up gear, less valuation at end of the competition.
(b) Miner's wages.
(c) Native labour.
(d) Air.
(e) Water.
(f) Drill Sharpening.
(g) Maintenance.
(h) Stores.

40. As regards the above items the following rules shall apply:—

(a) Machines and rig-up gear will be debited to cost at prices fixed as prescribed in para. 5, but will be valued by the Committee at the end of the competition, and the value so assessed will be taken to credit of the competing machine.

(b) A proportion of miner's wages will be charged, viz., 5s. per machine shift.

(c) Native labour will be charged at a uniform cost per head per shift on each mine throughout the competition.

(d) Air will be charged at a price per cubic foot according to pressure used, as fixed by the Committee. These prices will be uniform on all mines.

(e) All water used will be charged at cost.

(f) Drill sharpening and making drills will be charged at cost, which, if competitors supply their own sharpening machines, will include maintenance and depreciation, but not first cost, of sharpening machines.

(g) Maintenance—

  (i.) Spares, including hose and coupling, will be debited as issued; any returned unused being taken to credit.

  (ii.) Repairs mean actual shop charges and fitter's time charged at a standard rate fixed by the Committee.

(h) Stores, to include drills, lubricants and waste. Drills will be charged at ruling prices, except when supplied by the competitor, when they will be charged at prices fixed as prescribed in para. 5.

Sharpening.—41. Competitors may supply their own drill sharpening machines, but the Committee takes no responsibility for their safe return or for any damage sustained. Competitors may, with the consent of the Committee, instruct the drill sharpeners, except where otherwise arranged by the Committee. All drill sharpening will be done under the control of the Committee on the mine where the machine is competing, and by the usual drill sharpeners on the mine.

en decide which

Repairs.—42. Repairs must be done in the shops of the mine where each machine is competing, but competitors may provide their own fitter, who shall not be allowed underground.

F.—CHOICE OF MINERS.

Choice of Miners.—43. Miners will be chosen in pairs, to work day and night shifts in partnerships. Mine managers will be asked to nominate pairs of men, who should be machine men chosen on grounds of general intelligence, whether employed on the nominator's mine or not, and to state their qualifications and experience. From the list so obtained the number of partnerships required will be selected.

Conditions of Contract.—44. Miners will be paid 25s. per shift, with a bonus of one-eighth of any profit made by the Committee through the work in the stope in which the miner is employed. Each miner will sign a contract with the Committee to work the whole six months of the competition, and to permit the retention of 10 per cent. of his wages till the expiry of his contract. Should he leave before such expiry, except on account of illness or accident or other cause approved by the Committee, the amount retained shall be forfeited, and may be paid at the end of the competition to the miner taking his place. A miner temporarily absent through illness, accident or other good cause may be temporarily replaced by a man employed by the Committee on day pay.

Working Place.—45. Each partnership will as far as possible work in one stope throughout the trial, and will work in accordance with instructions given by the Committee.

Dismissal.—46. The Committee may dismiss and/or disqualify for a prize any miner whom it may consider to be incompetent or otherwise unfit for the position.

G.—BASIS OF AWARD FOR MINERS.

First Prize.—47. The first prize will be awarded to that partnership whose cost per fathom broken is least, provided that a factor shall be used to equalise the conditions in different stopes, such factor to be arrived at by the Committee, after obtaining an estimate of the price per fathom in each stope from a Sub-Committee consisting of the managers and mine foremen of all the mines selected for the competition.

Other Prizes.—48. The remaining prizes shall be awarded on the same basis.

Amount of Prizes.—49. The prizes shall be :—

1st —£300 to the winning partnership.
2nd—£200 to the second partnership.
3rd—£100 to to the third partnership.

Division of Prize Money.—50. In the event of a prize going to a partnership which has been changed during the competition, the prize money will be divided between the two partners working at the end of the competition according to the time worked by each, but any partner who has left on account of sickness shall receive a share proportionate to the time he has worked.

Stope Measurement.—51. The stopes will be specially surveyed at the beginning and end of the competition, and at such other time as the Committee may think necessary, on the lines laid down in the Interim Report of the Mining Regulations Commission.

Costs Charged to Miner.—52. The costs charged against the miners shall be the same as in the case of machines (see paragraph 39 above), with the addition of the cost of explosives used.

Withdrawals.—53. A deduction shall be made for fathomage estimated to have been broken by a miner with a machine, the entry of which is withdrawn or disqualified.

MEMBERS OF COMMITTEE.

Nominated by the Transvaal Government—

Mr. R. N. Kotzé, Government Mining Engineer (Chairman).
Mr. M. Fergusson, A.R.S.M., F.R.G.S., Deputy Inspector of Mines.
Mr. J. Orr, B.Sc. (Glasgow), M.I.Mech.E., Assoc.M.Inst.C.E., Professor of the Mechanics of Engineering, Transvaal University College.
Mr. J. A. Vaughan, M.I.Mech.E. (Lond.), M.Am.S.Mech.E., Chief Inspector of Machinery.
Mr. J. Yates, A.R.S.M., M.I.M.M., Professor of Mining, Transvaal University College.

Nominated by the Transvaal Chamber of Mines—

Mr. H. C. Behr, M.I.C.E., M.I.M.E., M.I.M.M., M.Am.S.M.E.
Mr. J. Blane, M.I.M.E.
Mr. F. A. Bristol, B.A.
Mr. T. J. Britten, M.I.Mech.E., A.M.I.C.E.
Mr. W. McC. Cameron, B.Sc., E.M.
Mr. D. Gilmour, A.M.I.C.E.
Mr. W. L. Honnold, B.Sc., E.M.
Mr. J. H. Johus, M.I.C.E., M.I.Mech.E., M.I.M.M., M.Am.I.M.E.
Mr. C. B. Kingston, B.A., B.A.Sc. (McGill), M.I.M.M., M.Am.I.M.E.
Mr. W. W. Mein.
Mr. S. Penlerick.
Mr. C. J. Price, M.Am.I.M.E.
Mr. A. M. Robeson, M.Am.I.M.E., M.I.M.M., M.Am.S.M.E.
Mr. L. T. Sicka.
Mr. J. W. H. Stubbs, E.M.
Mr. S. Thomson, E.M., M.Am.I.M.E., M.I.M.M.
Mr. E. J. Way, A.M.I.C.E., M.I.M.E., F.I.C. (Eng.), F.C.S., M.I.M.M., M.A.I.M.E.
Mr. G. E. Webber.
Mr. A. Yule.

as is generally supposed, as one will find if they stand close to the shaft at one of the levels when hoisting is in full swing. It is also said that in other countries the circular shaft is easier and cheaper to sink, but on these fields that point is open to great doubt, and I believe that in some parts of Scotland and the United States of America the rectangular is used on the coal fields in preference to the circular. Circular shafts were sunk for some little depth on the Langlaagte Royal and Primrose mines here. It would be very instructive at this point to know why they were not sunk to the depth originally intended. The costs, as compared to the rectangular, of same capacity, could, no doubt, be ascertained from the companies mentioned. Seeing that the circular shape has not been adopted in any of our later shafts, I take it that they were a failure.

I do not quite understand the author when he says—in referring to rail guides—"guides put in in this manner would prevent the use of a bend." I fail to see where a well laid out bend is any detriment to the use of steel rails; surely, the author has seen skips of all designs working around bends on steel rails here. Bends are bad things but sometimes they are unavoidable. In incline *v.* vertical shafts—for great depths—I fail to see where the advantages of the former over the latter come in. For incline shafts (*a*) it takes more footage to reach a given vertical depth; (*b*) consequently more timber, ropes, etc.; (*c*) the wear and tear on ropes is greater; (*d*) extra costs of rolls for rope; (*e*) heavier and closer timbering; (*f*) loading stations at every two or three levels at the least; whereas for vertical shafts (*a*) less footage; (*b*) less timber and ropes; (*c*) less wear and tear on ropes, guides, etc.; (*d*) no rolls; (*e*) lighter timbering; (*f*) loading stations can be placed farther apart; (*g*) quicker hoisting. Against the vertical shaft we have the extra cost of the cross cuts to intersect the reef, but then again an incline shaft is more subject to damage from subsidences, of which we have had several severe instances.

As regards the author's contention that too much timber has been used in our shafts, this to me, is a point which cannot be answered by a "yes" or "no." It is a pity that this point was not particularised, for unless one personally examines the shaft one cannot say whether too much or too little timber has been used. Neither can one say that too much timber has been used in any particular shaft simply because the one adjoining has used less, for although the formation here is pretty regular, yet in sinking, the strata in one shaft might be more broken and heavier than in the one adjoining, thus necessitating more and even heavier timber.

I am struck with the inconsistency of one of the author's suggestions. In one place he advocates the sinking of the two shafts in close proximity to each other to facilitate ventilation, development, etc., then farther on he puts down 7,000 ft. as about the distance apart. Surely, 7,000 ft. cannot be regarded as a short distance in mining; neither can I see where shafts so far apart facilitate ventilation or development.

Mr. Chas. B. Saner: The author, as usual, has presented the members of our Society with any amount of food for thought. He enunciates various theories, which, although different to one's own ideas, are difficult to controvert, and to discuss in detail would take too long.

I heartily agree that much can be learnt from other districts and industries and applied to our own; it is very foolish for anyone to crow from his own perch before finding out if there are any better roosters in the vicinity to knock him out. We can all learn by interchange of ideas, from the methods of others, and by obsrevation how not to do things and so improve on bad ideas. On the surface, given a modern plant, the running costs should not alter materially from month to month. But underground there are so many constantly varying factors, that a close comparison even between two adjoining mines is impossible, hence each mine should be judged upon its own merits. Even parts of the same mine differ greatly.

In some levels there may be only one narrow reef which is much faulted and cut up with dykes. In another section you may have two reefs close together, or a wide reef with good hanging and foot. All these conditions make a vast difference in working costs. Water may be cut, which at once upsets all calculations. Take the sinking of a vertical shaft, in good average ground speed is good, costs are low, and everything runs smoothly; then strike water or dyke or generally both together. Speed is greatly reduced sometimes by 50%; costs rise terribly and everything seems to get in a mess. As an example, I may mention that at the Turf mines shaft I was sinking for five months in hard dyke, which brought in some 80,000 gallons of water per 24 hours; this quantity is not great, but as it was continually following us down through fissures in the dyke we had to bale out every gallon from the shaft bottom and sink as well. Water is a curse when free, a blessing when under control. During the five months in the dyke the average sinking was 77·8 ft. per month at an average cost of £38 16s. per foot. During the last nine months, although we have passed through three small but dry dykes, our average has been 147 ft. and costs £26 5s., a reduction of £11 5s. per foot, but we had good bedded quartzites and the water caught up.

The author remarks "truly there are too many shafts, etc.," but in the early days our knowledge of the reefs and their extent was not what it is to-day, hence small properties and numerous shafts: as we come to learn more of these wonderful banket beds the laying out of properties is being considerably modified; fewer shafts are being sunk to attack larger areas. Look how much better the first row of deeps are laid out than the outcrops, and the deeper deeps than the first deep.

If it is quite sufficient to have only a 6 × 5 ft. shaft to haul 5,000 tons of rock or coal from 3,000 ft., how is it that in other countries the same terrible blunder is being made, as the author credits us with? From the last journals of the Institute of Mining Engineers, I see at the Bentley colliery, near Doncaster, two pits 20 ft. internal diameter were put down, designed to haul 2,000 tons up 600 ft. At the Polmaise collieries, near Stirling, two pits, 18 ft. × $6\frac{1}{2}$ ft. and $19\frac{1}{2}$ ft. × $6\frac{1}{2}$ ft. and two others $1\frac{3}{4}$ miles from these (9,240 ft.) 22 ft. × $6\frac{1}{2}$ ft., 16 ft. × $6\frac{1}{2}$ ft., were sunk about 500 ft. and 1,250 ft. respectively to haul not half the amount and from a much less depth. Owing to the reef conditions generally to haul the quantity stated by the author is, in my opinion, an impossibility. If all the rock were to be hauled from one level or station the shaft would naturally be much smaller, but with an incline reef and various levels to haul from, fast pulling is not possible. There is comparatively a small quantity of rock to be obtained from each level when it is worked out and abandoned. For instance, Kimberley as a comparison with the Rand for fast hauling is absurd, and you may as well compare Park station with Euston, as regards handling passenger traffic: when only hauling from one station money can be spent to advantage in making good bins, station and other facilities, but when a station is to be abandoned in a year or so, surely it is folly to spend money in elaborate arrangements.

At the Village Deep No. 1 shaft a large bin was put in at the bottom of the vertical into which the broken ore from three levels was dumped through "ore-passes" cut in the solid; this bin holds 800—1,000 tons. A "Kimberley" chute was put in at the bin mouth, which holds the exact quantity to fill a skip. By this means the skip is filled as quickly as the one above is emptied into the surface bin, and by means of an ingenious lip hinged on the skip, little or no fines fall into the sump and good fast hauling is done; and before the electric pumps were installed at the No. 2 shaft, water was hauled all night and the mill was fed from $1\frac{1}{2}$ levels; as well as all steel, tools, men and boys being handled. In giving us examples of fast hauling, details at the same time should be supplied of the conditions, width of seam or reef, dip, facilities of underground handling, etc.

It is strange that the author should accuse the Rand of being so far behind the front rank in mining methods when we have amongst us some of the best mining men in the world. We can easily haul the rock up the shaft, but the difficulty is to get the rock broken in sufficient quantities, and when broken get it to the shaft bins. Mining banket is not mining coal, and similar methods are not applicable to both.

I quite agree that the driver of an engine should be made comfortable while on duty; with an independent distance bell, not connected to the engine indicator, but one rung by the ascending skip or cage 80—100 ft. down the shaft, and a good headgear, giving plenty of room above the collar, accidents should be few or none with good drivers. Telephones underground are a great boon in large workings, and the old excuse that the banksman would immediately warn all that the boss is on the move is weak. Opinion is divided as to the best form or shape of shaft, whether round or rectangular, although I notice English collieries adopt both. Shape depends upon the formation sunk through.

I do not agree with the author that two shafts should be sunk close together; if dykes or faults or water are cut in one, the trouble will be in both. When the shafts are far apart, that is, dividing the property so that each shaft works half the area, there is no necessity to put the permanent pumps in each shaft. Equip one with good pumps and gravitate all the water to, is the best and present-day practice. Unless the ventilation is guided, the drives from the shaft will be no cooler than under the present system of initial development. With the pump and ladder compartments, or half the shaft well bratticed, excellent ventilation is obtained and can be guided into the development faces. Much faster development can be done by this method than with the two shafts close together, for at each level, four faces will be available against two. With the latest developments, arrangements are being made with neighbouring companies for each company to put down only one shaft; and the latest idea, which is excellent, is to erect all the necessary machinery, permanent headgear and engines at once, complete this, withdraw the construction staff and go ahead and sink at full speed. Until lately temporary hoist and headgear were put up, and sinking attempted under great disadvantages and delays while the permanent gear was erected over your head.

Cost per Ton should not be the be-all and end-all, but profit per ton should be the aim. By all means pack as much waste as possible underground; by this means safety is increased, and there is a saving in shovelling, tramming and hauling, and that is now being realised. If the new electrical heating furnace, at present being experimented with, proves a success, there will be a great saving in raising and lowering steel and in the quantity used; the great advantage of this invention being that an even temperature can be maintained, so a better class steel will be used with advantage.

Shaft Guides.—Two most interesting short papers appear in the Transactions, Institute of Mining Engineers, vol. 33, pp. 104-8, on various forms of shaft guides. Rope guides are much used in England, but have not found favour here, their place being taken by wooden guides, until comparatively lately, in section pitch pine 8 in. × 4 in. or 4 in. × 4 in. I prefer to have the joint between the dividers with a "backing piece," this "gives" a little and makes for smoother running. Steel rails of heavy section and slotted steel guides are now being put into the large shafts. These have the great advantage of giving great rigidity to the shaft timbers, and if rollers are put on the skip in place of steel shoes friction should be reduced and smooth running ensured. It would be most instructive to get comparative figures as to actual cost and life of the different styles. In one shaft I was connected with, wooden pitch pine guides were put in for sinking with coach screws. When the shaft was finished sinking these were taken out and 60 lb. rails put in; this was done by means of a 5-deck cage 35 ft. long, the corners of the compartment acting as guides; five white men on a single shift took out all the guides from two compartments and put in eight rails, or a total of 16,400 ft. in three weeks without in any way interfering with the work in the other compartments of the shaft. In another shaft three 8-hour shifts, each five men, were put on and 10,300 ft. of 60 lb. rails were placed in position and all wooden guides taken out and shaft examined and put in thorough order in 14 days. In the one case the job was contract, in the other day's pay and bonus. In putting down the steel slotted guides, my method is to bore the bolt holes in the dividers on the surface, and when a 30 ft. guide is needed, punch the guide on surface from measurements taken from the timber in position. We put a pair of guides in place by this means in half an hour. High lift electrical pumps in stations in the end of the shaft are doing good work in the latest sunk shafts.

Labour.—This is a subject I could enlarge upon at considerable length, but as my remarks have already assumed such considerable proportions I will only touch upon one or two points. To my mind, there is no better human muscular machine than the native, and in those parts where little brain is needed but steady work, he cannot be beaten, and it is impossible to replace three or four natives by one white man in the lower branches of mining. The European hewer of wood and drawer of water has little ambition, and is content to earn his small pay comfortably at home. If he comes to a foreign country he wants to rise. It is no good putting grown men in the mine, the ex-soldier, clerk or butcher, who cannot stand the dirt and rough work, and only takes it on as a very temporary billet; educate the youth, and there may be hope in years to come. The one great trouble with the native is his shortness of service; he is in a continual state of crude apprenticeship. I am in favour of establishing married quarters for natives, under proper control; I am sure by this means we should get a steady, contented and more efficient class of labour, and pay by results. Have well conducted eating houses, like mine boarding houses, and do no cooking in the compound. Let each native buy his food like the white man. If he will not work, then apply the motto: no work, no food.

Mr. E. M. Weston: One could easily write another paper of equal length, discussing all the points raised by the author, but I can deal only with a few points.

Our President has first already pointed out that it is quite a different matter to haul coal or rock from one station as compared with eight or ten. The author ignores the fact that owing to the laying out of the mine, due to faults, dykes, etc., it may be much cheaper, and indeed probably is in the cases the author has in mind, to employ two or three hoisting engines to haul 1,500 tons per 24 hours, than to employ one. I am glad to see the author has been reading up technical literature about controlling apparatus for prevention of overwinds. He has, however, forgotten to mention that designed by Futers, and in use in the Midlands. I am afraid the author will not convert many engineers to his idea of sinking circular shafts, especially if he were going to sink them without dividers. Personally, I would rather be out of such a shaft as soon as faulted ground or dykes were met with, and a good many miners would feel the same. It would be practically impossible to secure a circular shaft of large dimensions under such conditions, without the use of masonry or solid iron linings. If he is leaving dividers out, of course, bearers would be left out also. How is the author going to secure his rope guides while blasting in hard ground? The nice coal mining arrangements would soon be reposing on the bottom of the shaft. Bends in shafts certainly do not seem an unmixed blessing, and I think fewer will be put in in future. Is the author serious when he advises us to put in endless ropes on an incline shaft of 45°? Has he thought for a moment of the consequence of a stone dropping out on to the track and derailing only one truck for a start out of the hundred or so? How long does he estimate it would take to extricate and disentangle the final mess at the shaft bottom? Go easier Mr. Johnson, you must not startle us out of our sleep of rigid conservatism with such radical proposals. You may overdo the dose and the results might be serious. Why not put belt and bucket elevators in our vertical shafts! Nearly all the same advantages can be claimed for them as you claim for your proposal.

It is sad to see that all these engineers who have been worrying over dense air transmissions, electric transmissions, etc., have been wasting their time, and that the author has the true solution, which is, to take boilers and, I suppose, economisers and superheaters with feed pumps, etc., underground. See what a lot of pumping might be saved also, as the water could be lifted as exhaust steam!

If some of the mines, whose property is divided into two halves by a dyke or fault with 500 or 1,000 ft. throw, had been developed by the author's double-barrelled shaft they would be in an unhappy way now. Here again, I do not think the author serious. Has he never heard of bratticing a single shaft to make the mine as "cool and comfortable" as possible? Does he really contend that development can be done as rapidly from such a double-barrelled shaft as from two shafts set 2,000 or more feet apart and provided with brattices? "Much felt in practice," I should pity the author if he had to redeem a pledge to develop an equal or greater amount of ore in a given time by his scheme as could be done with two shafts in the proper position. How about future ventilation? Perhaps the author will recommend a system of bratticing the drives, and take out a patent for a rapid repairing machine to put them in again when they are blasted out or when the boy pokes a cocoapan through them. If one shaft only is to be sunk, I venture to put forward a scheme which should provide equal haulage capacity to the double-barrelled scheme with a lesser expenditure.

Briefly, it is this: a seven or eight compartment rectangular shaft, timbered with wood, would be sunk, but it would be about 2½ or 3 ft. longer than at present. Double dividers set 2 or 3 ft. apart would be put in between hauling compartments in the centre; the joints of wall plates would be put in between these dividers. While sinking, special hitches would be cut 50 ft. apart in this semi-compartment, and steel bearers of I channel beams would be put in with ordinary dividers at suitable intervals. The shaft would be sunk to the required depth with a temporary air brattice, and the only extra expense would be due to extra length of shaft, and the special bearers and extra timber—not big items. After the shaft was sunk 8 to 12 old hauling ropes would be hung from the collar set between dividers in the centre of the 2 to 3 ft. space. They would be spaced 9 in. to 12 in. apart and form a line across the shaft. They would pass between two I channel beams every 50 ft., and would be clamped in tension above and below each set, and perhaps between them. Then beginning at the bottom, the centre 2 ft. would be partitioned off on both sides and filled with concrete from wall to wall. This would not take long, as there would be skips on either side. For a 4,000 ft. shaft about 2,600 yards of concrete would be required; but a lot of large filling could be used; 1,600 ft. of I beams chains and some old ropes would be needed. The cost of all the work would be very little compared to that of another shaft. Mining from such a single shaft might then be permitted. The centre ferro-concrete wall would be of immense strength and would form a secure solid continuous centre bearer under the wall plates at their weakest place, rendering any collapse of the shaft due to heavy ground impossible. It would make a perfect fire-proof screen right from top to bottom. One shaft might be burnt right out without injuring the other. It would form a perfect airtight brattice, rendering the mine "cool and comfortable." Stations could be opened out separately for each half shaft, or the dividing wall built in on stations. It would offer immense resistance to any explosion such as occurred in the Driefontein shaft. Such an arrangement would make two shafts out of one and would avoid most of the objections against mining from one shaft. Even if an explosion wrecked the wall in one place, the end hauling compartments should escape damage. By making the concrete division larger a passage way for pipes and wires could be left in the solid of the wall, preserving them from water, and the experiment of concreting the main air pipe in might be worth trying.

Again, we wonder if the author is poking fun at us, when he suggests putting 12 machines on a 200 ft. face. First we would ask him, if it does not now pay to install motors on levels with 250 ft. of backs removing the ore slowly, how is it going to pay better to put them in for a shorter time for the same tonnage? I can imagine the state of mind a poor trammer would be in who had to keep such a face clean, when one of his trucks ran off the line and spilled. Why could drives be any further apart with this system than they are now? Surely, more rock must somehow be moved in a shorter time than is done now, and moved a longer distance. Why should brake roads, etc., help more than they do now, when they would have to be pulled down twice a day? What is our miner in charge of the face going to do if one lot of workmen drill much better than others? He must either sack the average man, or one lot of men would eat in ahead of all the rest, and the stope instead of assuming that ideal, so beautifully shown in the author's figure, would approach in shape the proverbial cow's hind leg. It would be quite a nice problem in transportation to get the dozen machines, bars, gear, hoses, pipes, etc., out of a 250 ft. stope on to the levels below or above twice a day; as obviously they would be scarcely safe in the stope itself anywhere. How long would the rigging up take next shift? One lot of men certainly are given a full shift of work which would take them the 12 hours, while the other men and the rigging gang have not enough to do. I am afraid even the policy of breaking rock until the stope was full of rock would not remove the objections to this scheme. If miners working two or more machines in a face are to break rock to greatest advantage, using their brains they must have lots of face to work on. Even three machines in a 160 ft. face are too many, and 80 ft. of face per machine pays every time. If the miners are to have good benches each time they set up, they must have room. There is more money lost to-day by the crowding of machines which is prevalent, than by not having enough at work.

I agree with the author that low costs are not always an evidence of good work, but I am afraid if some of his schemes were adopted we should have neither low costs nor good work. Speaking of grade, it certainly seems a peculiar method of keeping it normal, to spend the last week of the month hunting round the mine for any waste or unpayable reef to put in the mill to keep it down, if too high, or digging frantically at the rich stopes to whip it up, if too low. Such things are, I believe, known in certain high grade mines, and if you ask the mine captain, if anything below 16s. rock is left in the mine, he will tell you it is—sometimes.

I must now turn to the new system of placing stope holes. I am glad to see that the author believes in long holes. There are still mines not a hundred miles from here where the longest chisel that goes down the mine is 7 ft. With regard to the author's proposed arrangements of stoping holes, all I can say is, that, if a miner was working for me and I found holes like those shown struck in his stope, I should be obliged to sack him.

It would require several diagrams to go into this matter fully, and I have not time to draw them, but it can be easily shown that though the author's holes would, no doubt, break some ground, they are not efficient. The mistake most rock drillers make is shown very evidently in the author's diagrams. Most men have no idea or do not take the trouble to think what their first hole is going to break. In ninety cases out of the hundred it does not break in the crater form beloved of theoretical investigators, but tears most of the reef out in front of it. Nearly all the reef here breaks in slabs or heads at right angles to the walls, and in the author's case, were he to fire his front holes first and then go back and look he would find that the second hole had about a quarter of the burden on it should have, and the same with the other holes. Even in a stope of 38 in., if the top hole is given 24 in. burden, the bottom hole will break from 30 to 36 in. from the face. The author apparently and incompetent stopers place it vertically below the other hole, and probably give it the same charge of dynamite. As Prof. Yates has pointed out, it is this habit that renders machine stoping so much less efficient than hand stoping, and efficiency in machine stoping can only be brought up to a similar level in efficiency by using a sane imagination to judge just what each hole will leave for the next one to break. Every miner on going to a new stope should fire only one of his four or five holes first time, and go back even in the smoke to see just what it has done before he fires the other. It will be time well spent. When I saw the drawing of the author's new cut I said, now we have something at last that is new and original. Some friends of mine on contract at the Village Main Reef tried it. Their report to me reads somewhat as follows:—"We both made experiments with the new cut. We regret to say that owing to our limited financial resources and the necessity of breaking the round we have been temporarily obliged to discontinue them. Should, however, the author care to finance us for a few weeks and make us right with the mine captain, we will be pleased to resume them."

The author's remarks on the carrying of stopes appear to be based on experience. He recognises the fact that hammer boys are most useful in machine stopes. I have always contended that they should be used to do work in certain places where they are more efficient than machines. The author apparently thinks miners do not know their business when they tell him they object to using their machines under the bar in narrower stopes. If he thinks he knows better than those who have tried it, I am sure he is mistaken. I have tried it often enough myself, and it is an excellent method for wasting time, and should never be adopted unless in exceptional cases, where faults have thrown the reef down. I cannot agree with the author when he says that front holes should of necessity be given more burden than back holes, and that back holes must of necessity look in. If the bar is rigged in the right place there is no need for this in a four-hole bench in a large stope or a five hole bench in a small stope.

Surely it is the wrong kind of economy to try and use 2½d. steel. Unfortunately here high carbon steel at 5d. to 6d. a pound does not get proper treatment, as the author says; but such low grade steel, as the author recommends, is far too soft. The author's idea of rounding the edges of his chisel bits to give more cutting edge where it belongs, is an amusing instance of doing things the wrong way. What the author should have done was to make up the shoulders of his bit quite square and not rounded at all, if anything, somewhat concave. A section of the bit taken half an inch back from the edge of the bit should be in the form of a rectangle with sharp corners. Then the bit has one cutting edge parallel to the face of the hole, and four cutting edges parallel to the direction of the hole, to ream it out properly and cut the rock where there is most to cut. Taking out the centre of the bit is no new idea. Its effectiveness in chisel bits is spoiled by its weakening the cutting edge too much. The author will find that for this reason everyone using hollow steel tries to get his hole down on to the side of the cutting face. With star bits it is effective if it could be done economically, as thereby a core of rock is formed which is jarred off and breaks in large pieces. In Gillette's book on rock excavation the author will find particulars of the Derby tubular drill bit, designed to work on this principle with water feed.

Now, what do I really think about the author's paper? Well, I think the author determined to stir us up a bit, so he wrote a paper bringing the best arguments he could for all the bad practices in mining he could think of, with some useful notes from personal experience, hoping, as the French say, that he would at least "give us furiously to think," and perhaps evolve a few useful and new ideas. If this was his plan and motive, I am sure he deserves our best thanks.

SOME FEATURES OF SILVER ORE TREATMENT IN MEXICO.

By W. A. Caldecott, B.A., F.C.S. (Member).

Part II.—Theoretical Considerations.

DISCUSSION.

Dr. J. Moir (contributed): As regards this paper, I am mainly interested in the discussion by the author on the theory of the solution of silver sulphide in cyanide, more particularly the experiments described on p. 267. I wish to point out that the true equation for the reaction is more probably that given in the footnote of p. 266, simply because, in dilute solution, alkaline sulphides do not exist but are almost wholly hydrolysed into hydrosulphides. This also is borne out by the author's Table A, according to which the amount of sulphide solution required to give a coloration in argentocyanide solutions containing 16·9 molecules of free cyanide to one of silver, is directly proportional to the free cyanide present. The equation on p. 266 requires (applying the mass action law):
$[S''] \, [AgCy'_2]^2 = K[Cy']^4$ or since $[Cy'] = 16{\cdot}9 \, [AgCy'_2]$ in the experiment, $[S''] \propto [Cy']^2$; it follows that the equation governing this reaction is not the reverse of that on p. 266, but is $KSH + KAgCy_2 = 2KCy + AgSH$. The latter equation requires, as above, that
$[SH'] \, [AgCy'_2] = [Cy']^2$ which, in the author's experiment, reduces to $[SH'] \propto [Cy']$ which is just what he finds by experiment. It follows that the brown coloration is due to AgHS rather than to Ag_2S.

I have recalculated the author's figures, so as to allow for the effect of dilution due to addition of the reagent. It will be seen that the figures give quite a good constant at the end (see Table I. on opposite page).

The conclusion come to above is in agreement with earlier thermochemical work by Berthelot,* who found that the heat of neutralisation of dilute solutions of H_2S with one and two molecules of caustic potash were the same, which shows that the reaction $KSH + KOH = K_2S + H_2O$ does not occur in dilute solutions, whence it follows that when K_2S is dissolved it breaks up into $KOH + KSH$. He also found that no precipitate of Ag_2S, only a coloration, was obtained on adding H_2S to $KAgCy_2$ solutions, unless the solutions were made acid or boiled, which certainly suggests that the coloration is not Ag_2S but AgSH.

I have made some experiments to test this view: (1) Two different argentocyanide solutions were titrated with sulphide solution (*a*) alone, (*b*) after adding an excess of caustic alkali to convert SH′ into S″. In both cases the alkaline solution gave a precipitate with very little sulphide, the quantity required being the same for both: the quantity of sulphide was thus independent of the amount of silver present if there is an excess of alkali.

Table II.

No.	Cy′ Concentration.	$AgCy'_2$ Concentration.	S″ or SH′ Concentration	
A	·077	·004	·00023	without alkali.
B	·090	·009	·00010	
A′	·100	·005	·000011	with alkali.
B′	·100	·010	·000015	

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* *Comptes rend.*, 1899, pp. 630 and 706.

Table I.—Data of Table A of p. 267, recalculated.

Free Cy' % calculated as KCN in Original.	Silver in Original.	Na_2S used to first coloration. ·01 % Solution.	Corrected Volume of 100 c.c. Original	Cyanidion concentration in Equilibrium.	$AgCy'_2$ Concentration $Cy'/16{\cdot}9$.	Na_2S present in Corrected Volume.	S'' or SH' Concentration in Equilibrium.	Value of $\frac{[SH'][AgCy'_2]}{[Cy']^2} = K$
%	gm. litre.	c.c.	c.c.			mgm.		
·50	·490	2 65	105·3	·0728	·00431	·53	·0000645	·000052
·40	·392	2·20	104·4	·0588	·00348	·44	·0000540	·000054
·30	·294	1·55	103·1	·0446	·00264	·31	·0000385	·000051
·20	·196	1·00	102 0	·0301	·00178	·20	·0000252	·000049
·10	·098	0·50	101·0	·0152	·00090	·10	·0000127	·000049
·05	·049	0·35	100·7	·0076	·00045	·07	·0000089	·000069

One would expect from this result that excessive alkalinity would hinder the solution of silver ore in cyanide, even though the amount of sulphide in solution be not increased correspondingly, the action being due to the conversion of hydrosulphide into sulphide.

On the other hand, in concentrated solutions, it is probable that most of the sulphide is present as such (*i.e.*, not hydrolysed to SH'), in which case the first equation given by me holds good, and this appears to account for the precipitation of Ag_2S on diluting such a solution, as mentioned by the author on p. 268.

(2) *Experiments with Constant Cyanide and Varying Silver.*—These were made by weighing various quantities of solid $KAgCy_2$ into decinormal and centinormal cyanide solution and titrating with sulphide until a coloration appears after waiting for a minute. In the table the results have been corrected for change of volume by titration :—

Table III.

No.	Cyanide Concentration.	$AgCy'_2$ Concentration.	SH' Concentration	Value of K= $\frac{[SH'] [AgCy'_2]}{[Cy']^2}$
1	·047	·0047	·00003	·000064
2	·095	·019	·000025	·000054
3	·090	·009	·000057	·000063
4	·010	·0005	·000015	·000075
5	·010	·001	·000011	·00011
6	·010	·0024	·000009	·00021
7	·010	·0050	·000005	·00024

It will be seen that there is considerable deviation in the constant when the ratio between cyanide and silver is diminished.

The meeting then closed.

Contributions and Correspondence.

HONEYCOMB TUBE MILL LINERS.

With reference to my "Notes on Waihi Ore Treatment," read before the Society last year, I have received the following memoranda from Mr. H. P. Barry, superintendent of the Waihi G. M. Co., New Zealand :—

"The honeycomb tube mill liners continue to give every satisfaction. With castings 3½ in. deep, we get over 12 months' wear. By making castings slightly deeper and allowing the rough rock to project well above the castings, a life of 18 months could be obtained. Surely, there must be hard local rock at Johannesburg that would be suitable and that would come out much cheaper than cast iron or steel liners? It is of advantage to leave the rock projections very rough, as the rougher the liner the better, as it prevents slip of the mass of pebbles. We found out in the early days of tube milling that smooth iron or steel liners were bad on this account, so had ribs cast on them which prevented the slip.

There is no standard design for these liners; each man can make them the size and thickness he thinks best. It is well not to make the openings too large. I put eight segments into a 5 ft. mill, each segment being divided into four compartments."

Ralph Stokes.

May 30th, 1908.

DESCRIPTION OF AN APPLIANCE FOR WASHING SLIME OR SANDS RESIDUES.

The appliance described below has been used at the Simmer and Jack G. M. Co. for washing sands and slimes residues for the last two months and has given quite satisfactory results. The device is made from a piece of 8 in. pipe, 9 in. long, with a flange screwed on each end. The bottom end has a ¼ in. iron plate bolted on, through which a number of ¼ in. holes are drilled. Between this plate and the flange a piece of ¼ in. mesh iron screening and a filter cloth are inserted to form a filter bed. The top cover consists of a round piece of ¼ in. plate similar to the bottom cover, but in this case the bolt holes in both flange and cover are slotted out, so that the bolts can be removed by loosening the nuts, and the cover removed and replaced quickly. A ½ in. pipe is screwed into the cylinder immediately under the top flange and connected to an air main. The sample to be washed—sand or slime—is dumped into the cylinder and clear water added until nearly full; when the whole has been mixed up by hand the top cover is put on and the air valve opened slowly. The air entering at the top drives the liquid through the bottom filter, and after about three minutes the cylinder can be opened, when, if a slime sample has been operated upon, a cake containing about 15% moisture will be left on the filter cloth. This is again mixed up with clear water and the operation repeated, after which the sample is removed and sent to be assayed. Two washings have been found quite sufficient, and assays of the second wash filtrate only gave traces of gold. Cyanide managers will appreciate the advantage of getting a slime or sand residue washed in a few minutes instead of having to go through the tedious method of diluting with water, settling and decanting the liquid a number of times. If no air main is available an ordinary bicycle foot pump will supply the air pressure required.

Geo. O. Smart.

May, 1908.

NOTE ON BARBERTON "BLACK SANDS."

Looking through an old note-book recently, I came upon the following record of tests made upon samples of this product, which may be worth while placing upon record :—

In July, 1905, a sample submitted by Mr. Waites was examined, and it was found that from 83·3% to 84·5% could be separated by magneting. The original sample assayed 22·5 dwt. gold per ton, the magnetic portion only 1·5 dwt., and the non-magnetic residue went 131 dwt., showing that 94% of the gold became concentrated in about 16% of the weight of the parcel. In March, 1906, another sample assaying 5·5 dwt. was received from Mr. Harrison and tested with similar results, the magnetic portion removed representing 80·2% and assaying 1·0 dwt., and the non-magnetic residue 19·8% assaying 24 dwt., thus nearly 86·0% of the value was concentrated in 20% of the weight. Different parcels of this material, as is well known, vary greatly in value, but if the above samples represent a typical well concentrated product, the further concentration of roughly 90% of the values into 20% by weight, simply by the very cheap process of magnetic separation, obviously has a most important bearing on the extraction of the gold, lessening freight [illegible]res by 80% if, as usual, sent up to Johan[illegible]urg for treatment, and reducing by 80% also the tonnage to be submitted to the more expensive metallurgical treatment.

Personally, however, I think the best way of dealing with the problem would be the establishment of a custom works at some suitable spot in the De Kaap valley, in which case, if various estimates I have heard with regard to quantities available are reliable, a great revival of activity in the district should take place, and profits could be made in many cases which at present are quite unpayable.

G. H. Stanley.

Transvaal University College,
Johannesburg.

Obituary.

The following deaths are recorded with much regret :—

Mr. **Clement Lecky Johnson**, B.A. (Associate), who died on the 31st May, had been in very indifferent health for some years. He was connected with the General Mining and Finance Corporation for many years, having been for a considerable period the cyanide manager at the New Goch Gold Mines, Ltd., and for the last two or three years the caretaker at the Aurora West Gold Mine, at Maraisburg, where he passed away. Mr. Johnson was a B.A. of Dublin University, and in addition to his scientific attainments, he was an excellent cricketer, being a member of the first S.A. team, which visited England in 1893-4. Mr. Johnson was admitted an Associate of the Society in July, 1898.

Mr. **Francis Gore Knox Little**, who was elected a Member in Sept., 1906, and was for some time an assayer on the Crown Deep, Ltd., died some months ago in West Africa, whence he had gone to engage in mining pursuits.

Mr. **Harry Thomas Pitt**, who was elected a Member of the Society in Feb., 1898, died suddenly on the 2nd March from the result of an operation which had become necessary, owing to the after effects of a wound received during the war. Mr. Pitt was one of the oldest residents on the Rand, and was for many years cyanide and mill manager of the Rose Deep, Ltd., and was greatly esteemed by all with whom he came in contact. He served during the war with the Railway Pioneer Regiment, and was dangerously wounded at Sand River in 1900, from which he eventually succumbed. The funeral ceremony, at Braamfontein, was largely attended, and the Transvaal Town Police band and a firing party of the Witwatersrand Rifles, under Capt. C. B. Saner, being in attendance. The Society sent a wreath, and was represented at the funeral.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

PRODUCTION OF ALUMINIUM CARBIDE.—"The researches of the late Prof. Moissan in pyro-chemistry led to the discovery and isolation of many inorganic compounds, the production of which is only possible at the high temperatures attained by means of the electric furnace. Among the combinations thus effected for the first time was the interaction of carbon and aluminium to form aluminium carbide Al_4C_3. This carbide is distinguished from the carbides of other metals by the fact that it yields methane on treatment with water. Although earlier investigators had failed to obtain this substance in ordinary gas furnaces, Matignon has recently been led to repeat the attempt, owing to the circumstance that the heat of formation of aluminium carbide has been determined by Berthelot as having the high value of 245 major calories. The combination is accordingly a reaction accompanied by a great generation of heat, and should proceed to completion, if once initiated in a sufficiently intimate mixture of the two reagents. This theoretical view has now been confirmed by experiment. Finely divided aluminium and dry lamp black are intimately mixed with the aid of some turpentine, and the mixture, which should contain some excess of the metal, is heated for 20 minutes in a Perrot furnace, when the combination is complete. The excess of aluminium is extracted by rapidly washing in the cold with hydrochloric acid, or preferably aqueous caustic potash. The central portions of the fused mass consist of small hexagonal leaflets having a clear yellow tint, quite similar to the product obtained by Moissan. After removing aluminium, the carbide, when decomposed by water, gave rise to nearly 97% of the calculated amount of pure methane. The combination may also be effected by the thermite process without the application of external heat, and the various chlorides of carbon when heated with aluminium at comparatively low temperatures give rise to the carbide. With aluminium and hexachlorobenzene in sealed tubes the reaction takes place completely to form aluminium chloride and carbide, even at 225° C. Other chlorides of carbon interact when their vapours are lead over gently heated aluminium (*Comptes rendus de l'Académie des Sciences*, October 21, 1907). As a method for the commercial production of methane this reaction would not be so economical as the patented processes, in which carbon monoxide and hydrogen are caused to interact in the presence of finely divided nickel at about 230° C. Nevertheless, Matignon's method of obtaining carbides is capable of wide application."—*Times Engineering Supplement*, Jan. 8, 1908. (J. A. W.)

A NEW REDUCER IN FIRE ASSAYING.—"It is quite well known that much annoyance is experienced in assaying highly silicious ores by the contents of the crucible boiling over when argol or other carbonaceous reducing agents are used, and to avoid this it is necessary to start the fusion at a low temperature and heat gradually. (It was brought out in the discussion of this paper that a crucible is not so liable to boil over if the fusion is started in a very hot muffle.) Sometimes a cover of salt is used to prevent this boiling, but in that case the fumes are troublesome in pouring, if the slag is decanted from the lead.

This boiling is caused by the disengagement of carbon dioxide from the carbon in the reducing agent. (If a fusion is made without a reducing agent it will not boil over.) It occurred to me that if a sulphide were used as a reducing agent no gas whatever would be produced from this source, since the metal would combine with the silica and the sulphur be converted to sodium sulphate. This would have the double advantage of producing more quiet fusion and introducing a basic element into the slag. Probably the most convenient sulphide for this purpose is iron sulphide, which can be obtained free from gold and silver. This was tried with very satisfactory results.

Silicious ores that boiled badly, when a 0·5 assay ton was fused in a 20 grm. crucible, using argol as a reducing agent, gave no trouble whatever when the assay was made in a 15 grm. crucible, with the same quantity of ore, and iron sulphide used in the place of argol. According to the equation,

$$FeS + 4PbO = 4Pb + FeO + SO_2,$$

1 grm. of iron sulphide ought to reduce 9·4 grm. of lead. In practice about 9 grm. are obtained, so that the reaction is nearly quantitative.

The following charge was used: 0·5 assay ton of ore; 80 grm. litharge; 20 grm. sodium bicarbonate; 10 grm. potassium carbonate; 3 grm. iron sulphide.

I have never seen the use of iron sulphide as a reducing agent suggested in any literature on assaying, but it ought to appeal especially to those who assay sulphide ores by the nitre method. Its use instead of argol saves about 15 to 20 minutes in time of fusion."—C. A. ROSE.—*The Mining World*, Jan. 11, 1908, p. 61. (W. A. C.)

VOLUMETRIC DETERMINATION OF SODIUM MONOSULPHIDE.—"The volumetric method of determining sodium monosulphide which is based on its decomposition by the action of water into sodium hydrosulphide and sodium hydroxide, has the drawback that commercial sodium monosulphide almost invariably contains free sodium hydroxide. In the author's method this is obviated by first titrating the solution with N/10 sulphuric acid, with phenolphthalein as indicator, then adding formaldehyde solution in excess which acts upon the sodium hydrosulphide according to the equation—

$$NaHS + H.COH + H_2O = H_2COH.HS + NaOH,$$

and continuing the titration with the standard acid until the liquid again becomes colourless. The first titration gives the amount of free alkali in the salt together with that liberated by the action of the water, whilst the second gives the amount of sodium hydroxide formed by the action of the formaldehyde on the sodium hydrosulphide. Thus one molecular of sodium monosulphide yields two molecules of sodium hydroxide. Results thus obtained are stated to agree closely with those given by the iodine and zinc methods."—E. PODRESCHETNIKOFF, *Z. Farben-Ind.*, 1907, *6*, 388.—*Journal of the Society of Chemical Industry*, Jan. 15, 1908, p. 21. (A. W.)

ASSOCIATION OF ALUNITE WITH GOLD.—"In the Goldfield district, Nevada, alunite is so abundant and so generally associated with the intense alteration accompanying ore deposition that it must be looked upon as one of the most characteristic minerals of the gold deposits. This close association of the gold with a mineral which apparently had not been recognised in Nevada prior to the present investigation and which has been reported in comparatively few localities elsewhere, seems worthy of some description, especially as the relation promises to throw light upon the origin of the ores.

Attention was first directed to the occurrence of alunite in the district by the finding of small nests, or crystalline aggregates, up to 5 mm. in diameter, of a pale pink mineral in a greatly altered nearly white rock exposed on the east shoulder of Vindicator mountain, about 2 miles north-east of the town of Goldfield. The rock contains quartz phenocrysts and was probably originally a rhyolite. The microscope shows that the feldspar phenocrysts, as well as the ground-mass, have been altered to an aggregate of quartz and alunite.

With the alunite is associated a subordinate quantity of an anhedral, colourless mineral of high refractive index, strong birefringence and possessing a single well marked cleavage. The mineral is biaxial and optically negative, the plane of the optic axes coinciding with the cleavage. It resembles diaspore but differs from that mineral in being optically negative. It has not yet been identified. Undoubted diaspore has been found intergrown with alunite, in the altered dacite of the Florence mine.

Microscopic study of the bleached, altered rocks of the district has revealed alunite in nearly all of them. The rhyolite forming the summit of Columbia mountain, just north of the towns of Goldfield and Columbia, has been changed to a quartz alunite rock, the alunite, with some quartz, forming pseudomorphs after the feldspar phenocrysts which probably were originally orthoclase.

Alunite is very abundant in the altered andesite and dacite of Preble mountain and vicinity, and is present in practically all of the altered dacite which forms the country rock of the Combination, Florence, January, Red Top, Jumbo and other mines near Goldfield. It also occurs in abundance directly associated with the sulphide ores in these mines, in some places, particularly in the January mine, as a soft, easily pulverised material, of snow white or faint pink colour, which is not always distinguishable on mere inspection from certain pure forms of kaolinite which occurs in the mines of this and other districts.

At Goldfield, the delicate pink tint of the alunite is rather characteristic and serves often as a rough-and-ready means for distinguishing it from the kaolinite with which it is often associated. Close inspection of the altered rocks in which alunite is abundant usually reveals this faint pink colour in the aggregates which have replaced the original feldspar.

As a constituent of altered rocks in the Goldfield district, the alunite does not have complete crystal form. It is tabular in habit, the basal pinacoid being well developed and the rhombohedral faces being absent. In thin sections it shows distinct cleavage parallel to the basal planes and parallel extinction with reference to the cleavage. The interference colours in good thin sections are mostly yellow but rise in places to red or blue of the first order. The mineral gives a positive uni-axial interference figure, in sections showing no cleavage. The index of refraction is distinctly higher than the balsam of the slide (1·54).

The most decisive optical means of distinguishing the mineral from the optically negative sericite which it resembles in habit and refraction, is by means of the quartz wedge or gypsum plate. Sericite shows the higher interference colours when the section is turned with the cleavage at right angles to the direction of greatest elasticity (X) of the plate or wedge, while alunite shows the brighter colours when the basal cleavage is parallel to this direction.

The soft massive alunite, such as occurs in the January mine, when gently crushed on a microscopical slide and stirred in a drop of water, shows under a high power of the microscope thin, colourless scales of hexagonal outline. The average diameter of those examined is less than 0·05 mm., and such is their tenuity that no crystal faces can be recognised on the edges of the basal pinacoidal scales."—F. L. Ransome. — *Mining World*, Jan., 1908, p. 66. (H. A. W.)

Utilisation of Ceria. —"The production of pure thoria for incandescent mantles from monazite sands leads to an accumulation of large quantities of crude cerium salts as a bye-product for which at present there is no outlet. Attempts are being made to employ these compounds in the arts. Cerium salts have been tried as mordants in dyeing, and recently F. Gareli has experimented on the utilisation of cerium and its allies in the tanning industry. Hide powder rapidly decomposes the salts of cerium, lanthanum, and didymium, fixing these elements in the form of their hydrated oxides. These salts appear, therefore, to behave like those of aluminium or chromium. Ceric sulphate and the double nitrate of cerium and ammonium give rise to leather which has a pale yellow colour and resists the action of water. The leather obtained with thorium or zirconium nitrate is flexible and has a white tint. The hide partially reduces ceric salts and absorbs oxygen, which gives rise to a good quality of leather. The employment of ceric salts as oxidising agents in the manufacture of certain important chemical substances, such as benzaldehyde and related compounds, has been patented, and a series of cerous salts of the organic acids employed in pharmacy has been prepared with a view of ascertaining the therapeutic effect of cerium."—*Times Engineering Supplement*, Jan. 29, 1908. (J. A. W.)

Detection of Ruthenium in Platinum Ores. —"A portion of the alloy is melted with lead, the regulus treated with nitric acid, and the residue ignited in the air in order to expel the osmium. The residue, containing platinum, iridium, rhodium, and ruthenium is heated with a mixture of potassium nitrate and potassium hydroxide, and after cooling, the mass is extracted with water, and excess of nitric acid added to the solution, when the following reaction takes place:

$$2K_2RuO_4 + 4HNO_3 = RuO_4 + Ru(OH)_4 + 4KNO_3.$$

The brown liquid is poured into a beaker or test-tube, and the latter covered with a piece of filter-paper. If the alloy contained ruthenium, the paper will be blackened in the course of 12—24 hours, owing to the liberation of vapours of ruthenium tetroxide. It is stated that 0·01 gm. of ruthenium can be easily detected in this way. Care must be taken that the whole of the osmium is removed, as this gives a similar reaction."—N. A. Orlow, *Chem.-Zeit.*, 1908, *32*, 77.—*Journal of the Society of Chemical Industry*, Feb. 15, 1908, p. 125. (A. W.)

Detection of Formic Acid.—This depends on its reduction to formaldehyde by metallic magnesium. The solution to be examined is digested with the metal, in the form of ribbon or powder, at the ordinary temperature for some minutes, and the resulting product is tested for formaldehyde by any of the usual colour tests, or if the quantity is sufficient by means of phenyl hydrazin. Soluble metallic formates give a similar result but the reaction is much more delicate when the free acid is employed. H. J. H. Fenton and H. A. Sisson.—*Proceedings Cambridge Philosophical Society*, xiv., iv., 385. (J. A. W.)

GOLD LOST BY VOLATILISATION IN ROASTING. "As the author's general experience had not shown any marked difference between assays done on the raw or on the roasted ore, although the advantage, if any, lay with the latter, further experiments were tried to ascertain whether gold is actually lost by volatilisation during careful roasting.

In one experiment, a known weight of a rich mercuric telluride ore was placed in a combustion tube, the ends of which were loosely plugged with glass wool and asbestos to prevent any escape of particles of ore. One end of the tube was left open to the air, but the other end was attached by a rubber tube to an aspirator, there being a Woolf's bottle between the glass tube and the aspirator. The gaseous products from the tube were all drawn through water in this Woolf's bottle. The tube was placed in a combustion furnace and aspiration started so as to draw a current of air through the ore, and, after gradual heating to a cherry red, the ore and sublimate were assayed separately.

The portion of the tube containing the oxidised ore from which all tellurium and mercury had been distilled away, was cut off, ground up and assayed, tube and all, the portion of the tube containing the sublimed mercury and tellurium being also ground up and assayed, tube and all. The result was that no gold was found in the latter, while the former showed a gold content which agreed with the roasting assays done in the ordinary roasting dish with subsequent potting, etc., and also with the direct crucible assay.

The water through which the gaseous products containing the last traces of tellurium, etc., were passed, was tested for gold, but yielded none.

The result of this experiment led the authors to suspect that the losses often attributed to mercury in the assay of telluride and other ores were either much smaller than is often believed, or were non-existent.

To test this, 50 gm., of the same rich ore was taken and cinnabar ground up with it, so as to give it a 4% mercury content. The tube experiment was then repeated, with the result that a bead of mercury was obtained from the distillation weighing nearly 2 gm. The mercury, which was coated and much contaminated with tellurium, was removed and dissolved in dilute nitric acid, but no gold was obtained. The rest of the sublimate, which contained mercury, but was composed chiefly of tellurium and sulphur, was carefully scraped out, dissolved in aqua regia and tested for gold, but, as in the case of the mercury, no gold was found.

In addition to these special experiments, a great many assays by roasting in the ordinary open dish and subsequent potting in the usual way were made, precautions being taken in every case to prevent mechanical loss through excessive draft, or during stirring. In the case of the rich ore before mentioned, the results obtained were always comparable one with another, and, although slightly higher assay results have been obtained from this ore in one or two cases by other methods, the repetition of the particular method by which the higher result has been obtained has always failed to give again this high value, and the great majority of other methods, which the authors have tried, have given lower results. These few exceptionally high results, such as occasionally occur with all specially rich ores, can only be attributed to sampling, although the ore actually employed appeared to be perfectly uniform, and every precaution was taken in sampling.

The value of the ore by the first tube experiment worked out at gold, 0·8473%, that is, 247·130 oz. = 247 oz. 3 dwt. per ton, and the value of the same ore, taking the mean result of six very concordant open dish roasting assays, was 0·8463%, and again on four more concordant roasting assays, 0·8472%. This shows, in the first case, a difference of only about 6 dwt. per ton as compared with the 'tube' assay, and in the second practically no difference.

In the assay of five poorer telluride ores ranging in value from 1 to 3 oz. per ton, the results obtained from the roasting method have commonly been slightly higher for gold than by potting the ore direct, although generally somewhat lower for silver.

From these results, it may be taken as proved that the loss by direct volatilisation is very slight under proper conditions."—G. T. HOLLOWAY and L. E. B. PEARSE.—*Mining World*, Feb. 8, 1908, p. 252. (H. A. W.)

METALLURGY.

AGITATION OF SLIMED CONCENTRATES. — "The solution in the agitation vats is made up to 0·1% KCN and the pulp is agitated, on an average, for 36 hours, the stirrers revolving at a speed of 12 r.p.m.; after agitation the percentage of KCN falls to 0·064%, and the average consumption of cyanide amounts to 1·25 lb. per ton. Considerable trouble was experienced at the outset in dissolving the gold. The average time of agitation was 90 hours, and, although the resulting residues were good, the consumption of cyanide was high. This difficulty was overcome by the installation of a small air lift in each vat near the side to aerate the pulp. This method proved very effective, and brought the time of agitation down to an average of 36 hours, and decreased the consumption of cyanide from 2 lb. to 1·25 lb. per ton. The tonnages are arrived at by the specific gravity method, and check closely with the filter press tonnage. The extraction by cyanidation averages 96% in the agitators, but, taken on the raw concentrates, the extraction is equal to 81% of the original gold contents. The 16% extraction by amalgamation makes a total recovery of 97%. The residues, based on the raw concentrates treated, average for three months, 1·563 dwt. per long ton, or 1·395 dwt. per short ton."—*Journal of Chamber of Mines*, Western Australia, Sept. 30, 1907, p. 853. (W. A. C.)

DUST FROM ZINC GOLD SLIMES.—"It is universally recognised that there is a loss of values, by dusting, during the preparation of zinc-box sludges for smelting treatment. A hood, suction fan, and mechanical dust-collector system was installed in the clean-up room at the Great Boulder Perserverance mine at a cost of £60, and subsequently operated as required during the period from October 1, 1906, to June 8, 1907. All screening and mixing of sludges was done under the pyramidal-shaped hood, fitted with side curtains reaching to within a few inches of the floor level. By means of the fan the dust was drawn upwards, sucked away through an opening in the top of the hood, and delivered to the dust-collector, from which the recovered material was removed at the end of the period indicated. The quantity of dust recovered was 16·3 lb., containing 29·756 oz. of fine gold valued at £126·46. The total weight of sludge handled during the eight months period was 20,784 lb. and from this 46,790 oz. of fine gold were recovered by smelting. Thus the recovery of gold from the dust, though insignificant when compared with the total values handled, justified the cost of installation of the system."—*Journal of Chamber of Mines*, Western Australia, June 29, 1907, p. 727. (W. A. C.)

USES OF TITANIUM.—"Experiments with titanium in cast iron show that it adds greatly to the strength, and ferro-titanium is now being manufactured at a number of places in this country and in Europe for this purpose. It is said to add also greatly to the tensile strength and elastic limit of steel, and though much secrecy is maintained, it is known that titanium is used by various firms; and it seems probable that some steels, imported as containing vanadium, are really titanium steels.

If titanium will add to the good qualities of steel and cast iron, as its advocates claim, there would seem to be a large future for it, owing to its plentifulness and cheapness, in sharp contrast with the comparative rarity and high price of other metals used for the same purpose, such as vanadium, molybdenum, and tungsten.

Experiments are now being conducted by several firms in an effort to make incandescent electric lamps with titanium filaments. The problems connected with the reduction of titanium to a metal are so difficult, however, that the lamps have not yet been extensively placed on the market. The melting point of titanium is very high; private experiments are said to have shown it to be about 200° C. higher than tungsten, with a higher electrical resistance than that metal, so that its efficiency in an incandescent electric lamp is considerably higher.

Titanium as a component of rutile (TiO_2) and titaniferous magnetite is used in the electrodes of arc lamps, and titanium carbide is now coming into use for the same purpose. When one electrode is made of these substances, a block of carbon is used for the other.

Titanous chloride ($TiCl_3$) has been used to some extent in dyeing, as a mordant, but is said to have little, if any, advantage over stannous chloride for the purpose. Titanous sulphate $Ti_2(SO_4)_3$ is now being used as a stripper and mordant, and titanous potassium oxalate is used as a yellow dye and mordant in the dyeing of leather.

A new use for titanium as a detector of minute quantities of fluorine has been worked out by Mr. George Steiger, of the United States geological survey chemical laboratory. A solution of titanium sulphate $Ti(SO_4)_2$ is oxidised by hydrogen peroxide, and the degree of decolorisation of the resulting straw-coloured liquid by fluorine is used to determine the quantity of fluorine present. Very small quantities of fluorine can be determined in this way.

In porcelain tile, rutile is used to give a soft, beautiful yellow colour. It is also used to colour artificial teeth. For this last use, however, only the purer grades of rutile can be employed. The entire demand for titanium ore is still small."—FRANK L. HESS, *Mineral Resources of the United States.—London Mining Journal*, Jan. 4, 1908, p. 25. (A. R.)

USES OF ARSENIC.—"Arsenic trioxide As_2O_3 'white arsenic,' or 'arsenious acid,' is what is ordinarily meant by the general term of 'arsenic.' It is used as an insecticide in the preservation of hides and stuffed skins of animals and birds, in sheep and cattle dips and arsenic soap, and as a wood preservative, where it also prevents fungoid growth. It is used to prevent the growth of weeds and grass, in the making of some enamels, and many tons are used in the manufacture of glass. It is used as a fixer, or conveyor, of aniline colours in calico printing, and also enters into the manufacture of the dyes themselves. The use of arsenic as a poison for undesirable animals is familiar.

The greatest demand for arsenic is for making Paris green, used as a pigment, and as an insecticide on plants. Its toxic effect limits the former use. If pure, it is comparatively harmless to plant life, owing to its insolubility, but the trioxide is often present in some quantity, and if Paris green is then used largely it is fatal to plants, especially in dry climates, where it is not soon washed off by rains. Paris green is a copper aceto-arsenide, and should contain :—

	Per cent.
Arsenic trioxide (As_2O_3) ...	58·65
Copper oxide (CuO)	31·29
(C_2H_3O)	10·06

It is made by boiling verdigris, a mixture of basic acetates of copper, with arsenic trioxide. Scheele's green, a copper-hydrogen-arsenite, containing :—

	Per cent.
Arsenic trioxide (As_2O_3) ...	52·32
Copper oxide (CuO)	42·37
Water (H_2O)	4·81

is also used as an insecticide.

London purple, also much used as an insecticide, is a bye-product in the manufacture of aniline dyes, and is thought to be composed mainly of calcium arsenite with some colouring impurity mostly rosaniline arsenite. Besides London purple, there are two other preparations—Paris and English purple, which have practically the same composition. London purple analysed by the New Jersey experiment station gave this analysis :—

	Per cent.
Water (H_2O)	3·27
Arsenic trioxide (As_2O_3) ...	41·44
Lime (CaO)	24·32
Alumina Al_2O_3) Iron oxide (Fe_2O_3) ...	3·37
Sulphuric anhydride (SO_3) ...	0·31
Dye, by difference	27·97

It is made by boiling a waste product consisting of dye and arsenious oxide with lime to make the insoluble calcium arsenite.

Arsenic also enters into red, yellow, and grey colours. In medicine arsenic is used as a tonic, and it performs a similar office in various stock foods and medicines. Realgar burns with a fine white light, and is much used in pyrotechnics, either as the natural sulphide or as the artificial product made by fusing together arsenic and sulphur.

In metallurgy arsenic has but little use. It makes lead hard, and arsenical work from the silver-lead smelters is sometimes used for shot. The presence of arsenic, unless in minute quantity, in brass or bronze, makes it brittle, though small quantities of less than 0·5% give cleaner castings and a greater ductility to Muntz metal."—FRANK L. HESS, *Mineral Resources of the United States.—London Mining Journal*, Jan. 4, 1908, p. 25. (A. R.)

AIRLIFT AGITATION IN CYANIDING.—"That fine grinding of most of the ores to be treated by the cyanide process is beneficial is generally admitted. The quartzose envelope of the metallic particles has to be broken in order to allow contact with the dissolving medium and so, departing from former practice, it has more and more become the endeavor of metallurgists to 'slime' all the ore and, abandoning the separation of the real slimes from the fine sands, agitating both together. Mechanical stirrers assisted by centrifugal pumps were used with some success, but it was found that the presence of even fine sands made the operation uncertain, and costly.

An improved method of agitating mixtures of slimes and sands has been invented by M. F. C.

Brown, manager of the Komata Reef Gold Mines in New Zealand and introduced by him on several important works in that country with remarkable success. Compressed air is used for this purpose, applied in a peculiar manner in tanks. The tanks are cylindrical in appearance but consist internally of a cylindrical part terminating in a cone with a 60° slope. The weight of the tanks and contents is transmitted by the continuation of the cylindrical shell, stiffened by vertical angle bars, to a double angle iron resting upon a solid foundation. The lower end of the cone is closed by a cast iron bottom. A doorway in the lower part of the shell gives access to the space around the cone and a manhole in the latter facilitates inspection of the valves etc., when the tank is empty. The discharge of the contents, after agitation, takes place through a pipe near the bottom, provided with a valve, cock or plug.

The internal fittings of the tank consist of :—

1. A central tube called the airlift, of a diameter of about one-twelfth of that of the tank. It is open at both ends and extends from about 18 in. above the charging level to the same distance from the lowest point of the cone. It is supported by means of brackets of which there are three in each set.

2. An airpipe in the center of the lift, closed below and resting on the bottom of the tank, has at the level of the inlet of the airlift a 'sleeve' valve which prevents the pulp entering and allows the air to escape in an upward direction when the air-pressure exceeds that due to head of pulp in the tank.

3. A second airpipe outside the lift which serves to keep the pulp in gentle motion while filling or emptying the tank when the submergence is insufficient to allow the airlift to work.

4. An apparatus consisting of an annular casting surrounding the airlift, supported on the sides of the cone and provided with a number of pipes having sleeve valves discharging against the cone, the apparatus being connected to the air, and water or solution pipes. This arrangement is sometimes useful, when discharging, to wash down the sands which may have settled on the cone after the agitation is stopped.

The modus operandi, like the whole apparatus is very simple. The tank being filled with ground ore and solution, air is admitted through the pipe, which mixing with the contents of the lift, lightens the column inside and causes it to overflow, while fresh pulp runs in at the bottom and is in turn brought to the top, thus producing a perfect circulation which is kept up as long as the supply of air lasts. The initial air-pressure has to exceed that of the column in the airlift, but when once circulation has been established the pressure can be considerably reduced. At first also a greater quantity of air is required in order to obtain a sufficient velocity of circulation to remove all the pulp which may have packed on the cone during the period of filling. In commencing operations 50 lb. of air-pressure are used, but when the cone is clear of sand, the half of this pressure is sufficient. The quantity of free air consumed depends upon the proportion of slimes and sands, the degree of grinding and the viscosity of the pulp. In most cases 100 cub. ft. of free air per minute will keep the mass in lively motion and prevent the settling of sands on the cone. Large tanks, loaded with slimes only are agitated in New Zealand with an expenditure of less than 1 h.p. For a plant treating 100 tons per day of mixed sands and slimes a 10 h.p. compressor is sufficient.

The agitation by these means is perfect and very efficient aeration of the mass is obtained at the same time. Even after weeks of rest, when the contents of the tank have packed to a hard mass in the cone, agitation can be readily started, and after about one hour the mass is in perfect circulation. Sands can be treated as readily as slimes, and as they fall through the solution more rapidly than the slimes they receive a more energetic treatment. It is clear that the dissolving action of the cyanide solution is much more rapid than when less perfect modes of agitation and aeration are used. In fact, 24 hours' treatment are generally sufficient, and extractions of 92% of the silver and nearly all the gold have been obtained in that time.

The first cost of a plant of this description is much less than when mechanical agitation of the ordinary kind is used, and the treatment charges are also considerably reduced. One man can do all the work connected with the agitation of a 100-ton plant. There is no danger of anything getting out of order, and no chance of the contents settling down."—*Mexican Mining Journal*, Jan., 1908, p. 20. (A. R.)

PROGRESS IN GOLD-ORE TREATMENT DURING 1907.—" It will not be possible for me to review the progress of gold-ore treatment for 1907 in as detailed a manner as is usual, owing to these notes being written while travelling some thousands of miles away from my base.

Fine sliming and the treatment of slimes is again the most interesting feature of the year's progress—even former exponents of coarse crushing are now converts to, and profit-takers of, modern slimes practice. From Eastern Asia to Western Australia and from South Africa to North America sliming methods are pre-eminent and have modified the other methods or processes employed in the industry, such as crushing to tube mills and roasting ores ground to very fine particles.

Crushing.—We have heard less this year of 'Pans *v.* Tube Mills.' Experiments in the Transvaal appear to have resulted in favour of the latter, and many more tube mills have been installed. With the heavy stamps previously referred to in these notes the Luipaards Vlei appears to have been able to output 8·5 tons per head per diem, but in Rhodesia I am advised that 10 tons per head is being obtained in more than one instance—crushing, however, to pans instead of to tube mills. In Rhodesia the practice differs from that on the Rand ; with their much smaller installations pans are found to be more suitable, as being cheaper to efficiently install more convenient units and making less demand on power ; consequently we hear of but few tube mills operating in Rhodesia. The pans thus used are of the West Australian type.

But tube mills have unquestionably been improved in their *modus operandi.* Linings now last much longer, and instead of having to lay off a mill every three weeks or so for a week for a new lining, a mill can be run for three to six months with a lining placed in position in from two to three days. 'Automatic' linings such as those of El Oro, where pebbles are supposed to fix themselves in rebates cast in iron liners, do not yet appear to have proved themselves entirely satisfactory, and changes have recently been made in the shape of the recesses, which now have their sides chamfered to each other.

The use of quartz or other reef matter to be crushed in place of imported pebbles has much increased, and this and the utilisation of reef matter and local chert or other hard rock in linings of the Barry or Brown type have very materially reduced the cost of operating tube mills. In Africa a mill is started with a charge of pebbles containing, say, 10% of reef matter (stones), then in regular work it is found possible to reduce the daily feed of pebbles by 5%, adding reef matter (stones) to balance until the daily feed contains only, say, 10% of pebbles, the whole of the remainder being reef matter.

Slime Treatment.—It is in the treatment of the slimes produced by tube mills and otherwise that the progress of 1907 was most manifest. It may be regarded that the decantation process, which originated in Africa, and has so long survived there, has now had its day. Designed to treat material of low value, at that time not amenable to any method of treatment known to the industry, it resulted in profits being realised from material which otherwise must have run to waste. But the huge and expensive plant and low extraction had to give place to more modern automatic methods capable of cheaper equipment and higher saving, and already such well-known workers as J. R. Williams, W. A. Caldecott, E. J. Way, and H. S. Denny have been looking into methods employed elsewhere, notably the Ridgway, Butters, Burt and Merrill.

In my notes of last year reference was made to the new Denny plants at the Meyer and Charlton and the New Goch. The Denny brothers severed their connection with these two mines shortly afterwards, but in spite of their absence filter-pressing appears to be regarded as successful by the heads of the group, although much doubt has been expressed as to the wisdom of running cyanide solution through the mortar boxes. George Albu has criticised this as making it difficult to obtain the assay value of the mortar-box product, but I assume this point would not be regarded as serious if Mr. Albu were convinced he were obtaining a higher recovery of his gold. This discussion has been fully dealt with in another place.

On the Rand, however, the Denny methods have not been followed. Rhodesia on the contrary, having investigated the results, has had quite a Dehne filter-press boom, plant after plant having been installed during 1907.

Vacuum Filters.—But undoubtedly filter-pressing even with Dehne presses must give way before the suction filters of which so much has been heard in 1907. Cheap and efficient, the daily tonnage handled by vacuum filters is increasing by leaps and bounds.

Of the various vacuum filters on the market I must unhesitatingly refer first to the Ridgway. This differs from the filters of the basket type in being more rapid, working with thinner cakes, giving better washing and emptying itself (automatically) in more cleanly fashion, than the basket filters. Working as it does with the whole cycle of operation complete in 60 seconds on normal slime pulp, it will be seen that difficulties of keeping material on cloths during transit of frames or during emptying and filling of tanks, which arise in other methods, are entirely avoided, while the washing of so thin a layer (¼ to ⅜ in. thick) is much more rapid and complete than the washing of a cake of double or treble the thickness. The Ridgway is thus able to work with a much smaller area of filter cloth for any given capacity.

During 1907 a 500 ton-a-day plant was put into successful operation at the Great Boulder in West Australia, and I understand another plant of similar capacity has already been started. Plants of similar and smaller capacities have been erected (or are in course of erection) in Mexico, India, South Africa, Eastern Asia, etc., and altogether we hear much of the Ridgway filter at the present moment.

The official figures of the West Australian Chamber of Mines show Ridgway to be working under expensive local conditions (purchased power, dear labour and supplies) far under 4d. per ton treated ; in Africa the cost should be little more than half this.

Butters-Cassel v. Moore.—The Butters-Cassel filter has been installed very largely in Mexico as well as in the United States, not perhaps because it was the best vacuum type of filter—the recently published correspondence shows the pioneer Moore to be at least the equal and probably the superior—but because it was pushed by a man of repute and of great energy who was recommending the best thing he could get hold of.

But in the recent correspondence in the technical papers it appeared to me that the only persons writing in favour of the Butters-Cassel filter were those who were, or had formerly been, associated with Butters, while the Moore was recomended time and again by persons in no way associated with Moore, who had installed it after investigation and who were apparently in no way connected—not even by paying royalties—to Moore or his associates. Certainly if the Moore process had been run with half the energy, experience, knowledge and skill of the Butters I cannot imagine the latter type of filter to be employed at all.

For it is obvious that the hoisting of a basket of frames is a much neater and better and quicker expedient than the pumping in and out of pulp and solution or wash, each successive charge being mixed with the residue of the previous charge whether of pulp or of solution—if used—or of water, and it is within the knowledge of men experienced in slimes filtration that the maintenance of a vacuum in a basket half immersed—or partially immersed for, say, four periods of 10 minutes each during each cycle of operations—does not make for good results. The lower portion of the cake is being thickened by pulp deposit while the upper is cracking from the inrush of air, an unequal cake is formed and the washing must be unequal and there seems to be a very great waste of water in discharging the cake after filtration. From the published correspondence it seems to be laid down without serious contradiction that the Moore is more accessible and open to inspection, and that it does more work for a plant of given cost or in a given time.

Further modifications of the basket filter have appeared in the form of enclosed filters worked by pressure or vacuum, such as the Blaisdell, Burt, and Kelly. This form of filter is scarcely new, and when tried some years ago proved itself unhandy, liable to freeze or choke, difficult to wash thoroughly and difficult to discharge—in a word, it seems to be an attempt to work in the dark. Of course, modifications of the standard immovable inclosed type are made ; one runs the frame into a cylinder, or runs the cylinder away from the frame ; this makes the operation more accessible, but so far does not appear to be as successful as the pioneer basket type.

Merrill's Plant. — Reverting, however, to filter presses, Merrill has at last got his big plant at work after three years' labour. It appears now to be working to nearly its full capacity—according to my recent information—and to be treating slimes at a remarkably low rate. Advantage has been taken of every natural condition to make a successful working plant, and using static pressure to fill the presses and wash their charges, the Merrill costs are probably as low as any slimes treatment method yet put into practice. But my information is that with such a method the satisfactory treatment and hydraulic discharge of thick cakes in huge filter-presses is possible only with crystalline or granular slime pulp and with a great waste of water. In a word, it appears that successful results may not be anticipated by the process on ordinary slimes pulp in ordinary gold fields under ordinary economic conditions; the thorough washing of 4-in. cakes is feasible only with crystalline, granular or readily permeable slimes pulp.

The successful handling of slimes pulp has also caused more attention to the solution of the gold contained therein. Last year I referred to the system of treatment based on feeding solution or wash water at the bottom of a tank, the slime contents of which were in a state of gentle agitation, the idea being an overflow of clear solution containing the gold content of the charge. W. L. Holms of Mexico City (late of Consolidated Gold Fields of South Africa) brought this to my notice some years ago, but it did not appear to make headway. Bewick, Moreing & Co., were working it in 1906 in West Australia, but evidently without commercial success; in 1907 Adair Usher boomed an apparently similar process in South Africa. Possibly some metallurgist at the Geldenhuis Deep or elsewhere may make something tangible out of the idea, but it looks as though the only result would be to call attention to the extra extraction obtained by the increased agitation of the pulp, and in this connection the work of the Brown agitator ought to be investigated carefully.

The Brown Agitator.—This apparatus, introduced into New Zealand at the Komata Reefs, depends on the principle of the lessened specific gravity of a centre column into which air has been introduced at just such a pressure as will overcome the weight of the column of water at the point of introduction. There is no question of a jet of air circulating solution on the principle of the injector, but merely the physical lightening of the weight of a column by the displacement of a small proportion of the water by air.

Brown uses long narrow vertical (cylindrical) tanks of, say, 40 × 10 ft. or 55 × 15 ft. with a centre column of 1 in. diameter per 1 ft. of tank diameter. Into this central tube or column the air is introduced and agitation is so effective as to lift stones at, perhaps, the smallest horse-power per ton agitated of any efficient mechanical agitator. The power taken appears to be about 2½ h.p. per 50 tons (slime) pulp charge, which is the slime content of a 40 × 10 ft. tank. By this method the advantage of the accelerating action of air agitation in certain ores is obtained at a small cost.

The use of these tanks has been offered to the Waihi—the manager of which speaks very highly of these agitators—and Waihi Grand Junction in New Zealand, and a number have been installed in Mexico under the name of the Pachuca agitator.

Cyanidation of Silver-Gold Ores.—The progress of cyanide treatment for the silver-gold ores of Mexico has been one of the features of the slime boom. Chihuahua has been adopting cyanide with avidity, and Pachuca appears to be now coming into line. Of course El Oro has done pioneer work and Guanajuato has also been in the forefront.

In almost all instances the practice is similar, tube-mill sliming and treatment of the slime pulp at first by decantation, but now by basket or Ridgway filter. Very high extractions are claimed, but having regard to the natural refractoriness of silver sulphide ores—in some instances as at Chinacates associated with pyrites, chalcopyrite, galena and blende—one is impressed by the necessity of having recourse to the most modern practice, whether for getting the silver and gold into solution or for recovering the solution from the pulp for precipitation of the precious metal.

There is but little new either in roasting or concentration on gold ores this year. Merton and Edwards still hold the field for roasting, and the many flotation processes more or less (unfortunately with more of the less) successfully applied to the zinc-lead ores at Broken Hill, such as the Potter, Delprat, De Bavay, Cattermole and Elmore, do not yet appear to be successfully running at any gold mines known to me; although it is thought that the mechanical and metallurgical skill at the back of the Elmore process will bring it to the forefront during the coming year (1908)."—ALFRED JAMES.—*Engineering and Mining Journal*, Jan. 4, 1908, p. 17. (K. L. G.)

MINING.

MINING AND METALLURGY IN 1907.—"The study of the origin of ore deposits has a practical bearing on prospecting and on the valuation of mining properties. Theoretical studies should therefore be welcomed by practical men. It is agreed that the metals found in the earth's crust come in various ways from a nucleus, with a specific gravity exceeding 5.5, in the centre of the earth. This central nucleus may be compared to a bath of molten metals, the slags of which constitute the eruptive rocks. The magmatic segregation or igneous crystallisation which occurs within these eruptive rocks during the process of cooling is the cause of the formation of many ore deposits of value, including iron, manganese, chromium and nickel. The most faithful reproductions of ore deposits yet produced have been published as coloured photographs by Mr. B. Baumgartel. They were obtained in a most ingenious way. Directly a suitable fresh lode surface in the Harz mines was opened up, the author took a photograph of it with magnesium flashlight. The photograph was developed the next day, and on the day after the author again went underground and indicated with coloured pencils the distribution of the ores and lodestuff in the photograph, the face being illuminated for the purpose by a miner with an acetylene lamp. The final colouring was executed subsequently.

Observations on underground temperature have been made in the Pas-de-Calais coalfield. The temperature increases 1° C., not with every 31 metres, as has hitherto been thought, but only with every 56 metres. At a depth of 1,209 metres the temperature in a borehole was 35° to 40°. Various improvements in deep-boring appliances are recorded, and an electro-magnet has been successfully used in recovering a broken drill from the bottom of a borehole. Serious attention has been devoted to the surveying of boreholes, and several ingenious instruments have been devised for the purpose. In one of them the magnetic needle is clamped by the air pressure caused by the detonation of a cartridge. Mr. L. H. Cooke recently questioned the general reliability of borehole surveying based upon the use of the magnetic compass, on the ground of the polarisation of the boring rods by constant vibration.

Shaft sinking has received an unusually large amount of attention. Discussing special methods of shaft sinking, Prof. H. Louis stated that the largest amount of water successfully sunk through was at Horden colliery, where 9,250 gallons of water at a depth of 540 ft. were dealt with.

In explosives and blasting continued improvements are to be noted. On the occasion of the visit of the Iron and Steel Institute to the Styrian Iron Ore Mountain, which was ably described in a paper by Prof. H. Bauerman, a remarkable display of blasting took place. No less than 1,602 boreholes, of a total length of 6,050 ft., were loaded with 3,208 lb. of dynamite and fired simultaneously, bringing down 58,200 tons of ore.

The preservation of timber for use as mine supports continues to receive attention. Careful experiments made in the United States have shown that the most successful treatment, economically considered, is with creosote, the timber being immersed in an open tank, without pressure, in successive baths of hot and cold liquid. Reinforced concrete has been used to replace timber for roof supports, and success has attended the introduction of the Sommer pit props, made of weldless steel tubes, in Austria. On the Rand a strong preference for steel for constructional work is being manifested. Opinion in this country is less favourable to steel for underground requirements, owing to its shorter life underground as compared with iron.

Practical steps in the direction of winding by electricity have been taken at collieries in South Wales and at the mine of the Tarbrax Oil Co., Ltd., in Scotland. Experience tends to show that, compared with electricity, winding with the steam engine is cheaper in installation and in working costs, and consequently that when steam raising plant is available the steam winding engine is to be preferred. The use of turbo-generator sets is rapidly increasing, and Parsons' steam turbines have been installed at various collieries.

In ore dressing the improvements introduced during the year have been in details rather than in principle. The steam stamps have been discontinued everywhere except at the Lake Superior copper mines. The Harz jig continues to hold its own, and for fine concentration various modifications of the Wilfley table have been introduced. Magnetic concentration is making steady advances, considerable attention having been given to it for the separation of tin wolfram ores. Excellent results are being obtained in Scandinavia with the briquetting of iron ore. In gold extraction the use of tube mills has extended, leading to better returns from producing mines, though a careful computation of added capital, charges, and working costs in every case is necessary before their installation can be justified.

In metallurgical matters the principal changes seem to have been mainly in the direction of modifying accessory details in existing processes. The application of electricity to driving reversing rolling mills in Austria, and Mr. Sandberg's method of manufacturing silicon steel rails are the most striking novelties to record. Considerable progress has been made in the electric smelting of iron, and notices published in these columns from time to time during the year have recorded the results of successful experiments in various parts of the world. Aluminium is now manufactured by electric methods at a number of works controlling over 84,000 h.p., and there are some 34 electrolytic copper refineries at work in Europe and America. The process of pyritic smelting, where the sulphur in the ore is utilised as fuel in the blast furnace in the treatment of ores containing precious metals but poor in copper, continues to make way in districts where fuel is expensive. Hot blast has been abandoned at Mount Lyell, Tasmania, after four years' trial. It has also been given up at Ducktown, Tennessee, and at the Copper Queen mine, Arizona. The smelting of antimony ore has received less attention; and in lead smelting there has been a tendency to revert to older methods. The reinforced concrete dust chambers have fallen into disfavour, and the lime roasting process does not appear to have been an unqualified success.

The greatest honour conferred on the mining industry during the year was, however, the institution by the King of a new medal, for bravery in mines, to be known as the Edward medal."—*London Mining Journal*, Dec. 28, 1907, p. 793. (A. R.)

MISCELLANEOUS.

THE INTERACTION BETWEEN MINERALS AND WATER SOLUTIONS.—"The natural silicates precipitate the metals from solutions of salts, while at the same time the bases of the silicates are dissolved in quantities nearly equivalent to the precipitated metals. The bases most commonly replacing the metals in these processes are potassium, sodium, magnesium, and calcium. Where exact equivalence is wanting, it is attributable either to solubility of the mineral in pure water or to the precipitation of basic salts.

The metals are precipitated as hydroxides or basic salts (in the case of cupric sulphate, for instance, as a basic sulphate similar to brochantite or langite) with more or less metal silicate.

The specific materials on which work was done are albite, amphibole, augite, biotite, enstatite, garnet, clay gouge, kaolin, microcline, muscovite, olivine, orthoclase (3), prehnite, shale, talc, tourmaline, and vesuvianite, with cupric sulphate solution; and orthoclase with salts of sodium, potassium, magnesium, calcium, strontium, barium, manganese, iron, nickel, copper, zinc, silver, gold, and lead. Experiments were also made on the action of kaolin on solutions of salts of zinc and iron, and of glass, fluorite, and pyrite on cupric sulphate, and of carbonic and sulphuric acids on orthoclase.

Absorption, a mechanical surface attraction, plays a comparatively insignificant part, if any, in the retention of copper by kaolin. If absorption is lacking in the case of kaolin, it seems reasonable to assume that it is lacking in the case of other silicates also.

Under the conditions of experiments described the precipitation of copper caused by orthoclase does not differ materially in quantity from that caused by pyrites.

A dilute salt solution decomposes orthoclase and dissolves its constituents to about the same extent as does a saturated solution of carbonic acid. Both are much more efficient in this respect than water.

Orthoclase (and presumably other akali silicates also) accelerates the oxidation of ferrous sulphate by the oxygen of the air. A similar influence may be expected in other oxidation reactions that take place more readily in the presence of alkali than in the presence of acid.

It is worthy of note that the iron of ferrous silicates is not found in solution replacing copper after the precipitation of the latter. This was also true of the iron in a clay gouge containing considerable ferrous carbonate."—EUGENE C. SULLIVAN, U.S. Geological Survey, Bulletin No. 312.—*London Mining Journal*, Dec. 28, 1907, p. 783. (A. R.)

THERAPEUTIC APPLICATION OF ARSENIC COMPOUNDS.—"Numberless experiments have been made with the view of obtaining arsenic compounds, which, while giving the beneficial effects of this element, would be more readily tolerated by the system than either arsenious oxide or Fowler's solution (potassium arsenite). Sodium dimethylarsinate has been suggested in this connection, but its use is attended with one grave disadvantage—namely, the tendency which this salt possesses of giving rise to the very malodorous cacodyl compounds. Moreover, this substance is nearly as poisonous as arsenious acid itself. A much more satisfactory compound has recently been employed under the name of atoxyl. This substance, which is prepared by heating together aniline and arsenic acid at 200° C., was formally supposed to be the anilide of arsenic acid. Its extensive use in therapeutics has, however, led to the revision of its chemical constitution, and Ehrlich and Bertheim have recently shown that the compound is in reality a new organic acid—arsanilic acid, $NH_2.C_6H_4.AsO(OH)_2$. The drug is usually exhibited in the form of its sodium salt

$$NH_2.C_6H_4.AsO(OH).ONa,4H_2O,$$

which is readily soluble in water, giving a neutral solution (*Berichte der deutschen chemischen Gesellschaft*, July 20, 1907). Forty times as much arsenic can be administered in the form of atoxyl as can be tolerated in the condition of Fowler's solution. The drug is now being used for a variety of maladies, and has given very promising results in cases of sleeping sickness."—*Times Engineering Supplement*, Dec. 18, 1907. (J. A. W.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

MINES AND MINERALS OF THE BRITISH EMPIRE. By RALPH STOKES, late Mining Editor, *Rand Daily Mail*, Central News Agency, Johannesburg, 21s. net. (London: Edward Arnold.)

This book is a description of the historical, physical and industrial features of the principal centres of mineral production in the British Dominions beyond the seas. It is illustrated by a large number of excellent photographs, and fortified by several well digested and clearly presented groups of statistics. Undoubtedly, the keynote of this extremely interesting account of the mining activities of the Empire is revealed in the care, with which its author has investigated the labour conditions in all the districts covered by his tour of 35,000 miles. In this country, and at this time, no work on mining will better repay a careful study than this collection of the facts of mining conditions and their action on the problems of social economics. It is fitting, therefore, that the first chapter is devoted entirely to mine labour, and the various elements employed from Sudbury, Ontario, to Waihi, in New Zealand. Contrasts of great economic importance are revealed in the study of so wide a field, and the difficulty and complexity of the questions involved are treated with a frank thoroughness which renders this chapter as valuable as any of the following twenty-five.

Very many of the mines visited and clearly brought to our notice deal with other than the precious metals, and mention may be made of the vivid descriptions of tin working in the Malay States, nickel in Ontario, copper in Tasmania, and graphite in Ceylon. Lead, zinc, mica, asbestos and precious stones are all represented in these pages, and nearly every part of the Empire has been laid under contribution. The graphic accounts of the districts visited bring their features distinctly before the reader, and it is a further illustration of the modern shrinkage of world space, that so much has been seen and such lengthy journeys have been undertaken in so short a time. As a consequence, the result is a work completely up to date and, therefore, of exceptional value to the mining engineer, the investor and the student. It is worthy of a place on any bookshelf, and has so enhanced the reputation of its author as to engender a desire for many successors. (H. A. W.)

THE METALLURGY OF IRON AND STEEL. By BRADLEY STOUGHTON. 9 × 6, viii. and 509 pp. $3, post paid. (New York: Hill Publishing Co.)

As stated in the preface, this book covers every branch of the metallurgy of iron and steel, but the attempt to deal fittingly with such a vast subject in a limited space does not appear to have resulted altogether satisfactorily.

The great fault appears to be a want of proper balance in the treatment of the various sections; certain portions are so curtailed as to be quite elementary in character, while others are of real use to students and technical men; for instance, the construction of the blast furnace itself receives about nine lines! This and the various blast furnace accessories take up two and a half pages of type, while rolls are dealt with in detail in about twenty pages, and iron and steel founding in fifty-five pages.

Further, want of care in compilation is seemingly shown in several erroneous statements, notably with regard to Sheffield practice in crucible cast steel manufacture, the explanation of the heating of metal in the Kjellin furnace, and the nomenclature of silicates.

In order to make the rest of the book intelligible to those without the necessary knowledge of chemistry and physics, a brief section of about 30 pp. is introduced to supply the defects, in these directions, in the education of readers, and if thoroughly mastered would, no doubt, go a long way towards this end; whether this is likely to be the case, however, is open to serious doubt, and to properly understand anything of modern metallurgy a really good grounding in these sciences is essential. This section is marred by one or two unfortunate examples also.

On the other hand, the principles of the manufacture of pig iron and its purification, and the various methods of manufacturing steel are very well treated, as also is the mechanical treatment of steel and iron and steel founding; while the chapters on the solution theory of iron and steel, the constitution of steel and cast iron, and the heat treatment of steel are really admirable; if only for these alone the book is well worth reading, bringing, as it does, this constantly widening part of the subject right up to date.

The value of the book is largely increased by useful bibliographies at the end of the chapters, and it is copiously and excellently illustrated and well printed, as indeed are all those produced by this firm.

It is disappointing, however, that what is, in the main, an excellent book, should be marred in the directions noted, and scarcely, therefore, does justice to the reputation of its author. (G.H.S.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 142/08. Alexander John Arbuckle (1), Alfred Osborne (2). Improvements in continuous filter presses. 15.4.08.

(C.) 143/08. George Thomas Plumb. Improvements in signalling systems for mines and the like. 18.4.08.

(C.) 144/08. Alfred Henry McNeil. Improvements in electric fire alarms and thermo indicators. 18.4.08.

(P.) 145/08. H. C. Marshall. New water elevator. 18.4.08.

(P.) 146/08. S. S. Osborn. Improvements in amalgamators. 18.4.08.

(C.) 148/08. Horace George Nichols. Improvements in or relating to the separation of the liquid and solid components of a fluid, such as wet slimes or ore pulp or for the like or for analogous purposes. 23.4.08.

(C.) 149/08. Frederick John Cox. Improvements in or relating to carburetters. 24.4.08.

(C.) 150/08. Henri Herrenschmidt. Improved process of manufacturing bodies capable of conducting electricity in a cold state by using second class conducting bodies. 24.4.08.

(P.) 152/08. Alexander John Arbuckle (1), Alfred Osborne (2). Improvements in filtering apparatus. 25.4.08.

(P.) 153/08. Thomas Passe. Improvements in means for imparting the intermittent rotary motion to the bits or steels of rock drilling machines. 27.4.08.

(P.) 154/08. Thomas Passe. Improvements in and in connection with rock drills and like machines. 27.4.08.

(P.) 155/08. Robert Henry Copeland. A new or improved electrical process for the smelting of gold slimes and apparatus therefor. 27.4.08.

(C.) 156/08. Maximilian Ogilvie Tillard. Improvements in loading-boxes for inclined mine shafts. 27.4.08.

(P.) 157/08. William Archibald Shearer. Improvements in acetylene gas lamps. 30.4.08.

(P.) 159/08. James Garvie (1), Sydney Arthur Cole (2). Improvements in valve motions. 30.4.08.

(P.) 160/08. Frederick Corbishley. Improved sliding and folding partition for dividing rooms and buildings and filling in openings of same. 30.4.08.

(P.) 161/08. John Craig. Apparatus for treating diamond bearing deposits, ores and the like with liquids. 1.5.08.

(C.) 162/08. Camille Bornet. Improvements in percussion tools for mines and quarries. 1.5.08.

(P.) 163/08. Enoch Kenyon. An improved combined mercury trap and concentrator for use in the process of recovering metals from their ores. 1.5.08.

(P.) 164/08. William Thomas. Improved catch for skips in vertical or compound shafts. 2.5.08.

(P.) 165/08. Frederick Victor William Swanton (1), Charles Bramley (2). A new and improved process for the production of power by means of the combined action of air or gas and water and other fluid upon the same. 2.5.08.

(P.) 166/08. Kenneth Somerset Bell. Improvements relating to the supply of fluids to bore holes. 2.5.08.

(P.) 167/08. George Ritchie Lambie. Improvements in furnace fire bars. 2.5.08.

(P.) 168/08. Harry August Stockman. Improvements in the means for reducing the temperature of air or gas admitted at the intake of compressing plants: also applicable for creating an artificial down draft to displace foul air from shafts or the like or for ventilating purposes in buildings and the like. 6.5.08.

(P.) 169/08. James Spry. Improvements in safety brake mechanism for mine skips, cages and the like. 7.5.08.

(P.) 170/08. William Edward Wyeth. Process for obtaining carbonate of lead (commercially called white lead) direct from the ore as contained in mountain limestone and sandstone. 7.5.08.

(C.) 171/08. William Hooker. An automatic apparatus for supplying air for various purposes. 8.5.08.

(P.) 172/08. William Hooker. An improved primary electrical battery. 8.5.08.

(C) 173/08. Ernest Schonberg. Improved means for preventing damage or breakage at the working parts of grinding or crushing machines. 8.5.08.

(P.) 174/08. Hugh Young Glen. Improvements in explosives. 11.5.98.

(P.) 175/08. Hugh Young Glen. Improvements in explosives. 11/5/08.

(P.) 176/08. Robert McConochie Brown (1), John Dillon (2). Improvements in driving means for vibrating or shaking and concentrating tables and the like. 11.5.08.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

Brooks, E., *l/o* Krugersdorp; Golden Valley Mine, Gatooma, Rhodesia.

Chapman, F. T., *l/o* Germiston; Geduld Proprietary Mines, Ltd., P. O. Box 41, Springs.

Daniels, I. C., *l/o* Bembesi; Globe and Phœnix G. M. Co., Ltd., Queque, Rhodesia.

Dunning, F. G., *l/o* Johannesburg; Knights Deep, Ltd., P. O. Box 143, Germiston.

Leng, R. W., *l/o* Johannesburg; 30, Primrose Hill, Skinningrove, Carlin How, R.S.O., England.

Michaux, D., *l/o* Krugersdorp; Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston.

Milligan, J. T., *to* 24, Ascot Road, Bertrams, Johannesburg.

Moir, Dr. J., *l/o* Johannesburg; c/o Dr. Bruce, Dingwall, Ross-shire, Scotland.

Mosenthal, B. P., *l/o* Langlaagte; c/o Mr. T. A. White, P. O. Box 192, Germiston.

Robinson, L. J., *l/o* London; The Old Langham, Kerk Street, Johannesburg.

Woodburn, J. A., *l/o* Boksburg; c/o Mr. H. Taylor, P. O. Box 790, Johannesburg.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

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VOL. VIII. JUNE, 1908. No. 12.

Proceedings
AT
Annual General Meeting, June 20, 1908.

The Twelfth Annual General Meeting of the Society was held in the Chamber of Mines, on Saturday, May 16th, Prof. J. Yates (President), in the chair. There were also present :—

60 Members : Dr. D. Macaulay, Messrs. T. Lane Carter, F. F. Alexander, R. G. Bevington, W. R. Dowling, K. L. Graham, A. McA. Johnston, A. Richardson, G. O. Smart, Prof. G. H. Stanley, H A. White, Prof. J. A. Wilkinson, W. A. Caldecott, W. Cullen, R. W. Atkin, S. G. Bartlett, W. Beaver, F. W. Cindel, M. H. Coombe, G. A. Darling, A. D. Gilmour, J. Gray, F. G. Guthrie, A. G. Hoyer, M. Inglis, Dr. L. G. Irvine, H. R. Jolly, J. A. Jones, E. J. Laschinger, G. A. Lawson, Hy. Lea, Jas. Lea, W. D'A. Lloyd, W. P. O. Macqueen, G. Melvill, J. E. Metcalf, H. Meyer, J. T. Milligan, A. Mitchell, P. T. Morrisby, F. B. Ogle, Prof. J. Orr, J. F. Pyles, E. T. Rand, W. H. Roe, O. D. Ross, A. Salkinson, W. Sharp, F. L. Simmons, S. H. Steels, R. Stokes, W. H. Stout, J. A. Taylor, W. Taylor, J. E. Thomas, A. D. Viney, J. F. Walker and J. P. Ward.

15 Associates : Messrs. H. G. Brickhill, W. M. Coulter, J. Cronin, C. L. Dewar, A. L. Edwards, J. H. Harris, C. B. Hilliard, H. A. Owen, H. B. Powter, B. Southwell, A. Thomas, W. E. Thorpe, C. Toombs, W. Waters, and L. J. Wilmoth.

14 Visitors and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting as published in the *Journal*, were confirmed.

NEW MEMBERS.

Messrs. W. Beaver and W. Taylor were elected scrutineers, and after their scrutiny of the ballot papers, the Chairman announced that the candidates for membership had been duly elected, as follows :—

BROWNING, EDWARD, Robinson G. M. Co., Ltd., P. O. Box 1024, Johannesburg. Mining Engineer.

COCKBURN, PAUL ALEXANDER, Gaika Mine, Que-que, Rhodesia.

CULLEN, MATTHEW, Cyldesdale Collieries, Ltd., Wolvehoek, O.R.C. Colliery Manager.

HARRIS, J. H., P. O. Box 30, Cleveland, Transvaal. Amalgamator. *(Transfer from Associate Roll.)*

MACDONALD, PETER ARCHIBALD, Etna Development Co., Ltd., Gatooma, Rhodesia. Mill and Cyanide Manager.

WILMOTH, LESLIE JAMES, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston, Transvaal. Assayer. *(Transfer from Associate Roll.)*

WAY, EDWARD JOHN, Kleinfontein Group, Benoni, Transvaal. Consulting Engineer.

WESTON, EUSTACE MORIARTY, Rand Collieries, Ltd., P. O. Box 13, Brakpan, Transvaal. Mine Manager *(Transfer from Associate Roll.)*

The Secretary announced that the following gentlemen had been admitted as Associates by the Council since the last general meeting :

POWRIE, DONALD E., Rose Deep, Ltd., P. O. Box 6, Germiston, Transvaal.

PRICE, WALTER SAMUEL VINCENT, P. O. Box 1001, Johannesburg. Miner.

REECE, FREDERICK BENNETT, Homestake Mining Co., Ltd., "The Smead," Lead, S. Dakota, U.S.A. Assistant, Refinery Department.

WHELER, AUBREY STUART, c/o S. W. Jameson, Esq., 52, Ashley Gardens, Westminster, London, S.W. Mining Engineer.

The President called upon the Secretary to read the Council's Annual Report and the Statement of Accounts.

ANNUAL REPORT OF COUNCIL.

Your Council has much pleasure in submitting its Report upon the work of the past year, together with a Statement of Accounts and Balance Sheet for the twelve months ending the 1st June, 1908.

The depression, alluded to in the last Report, has not yet passed away, and its results are again reflected in the accounts before you. It is only by the exercise of much care and attention to detail that the Society is still in an excellent financial position. The revenue for the year shows a decrease of £33 2s. 2d., a decrease which would have been much greater but for the unremitting efforts of the Finance Committee in successfully dealing with the outstanding subscriptions, although at the cost of a great deal of extra correspondence and appeals to the members in arrear. In this connection it is necessary to point out, that according to the rules all subscriptions become due on the 1st July in each year, and are payable in advance, and your Council would earnestly impress upon all the desirability for compliance with that rule. The Society does not, as yet, enjoy any revenue from invested funds, such as is the case with the majority of Home Societies, and the non-payment of subscriptions on the due date renders it extremely difficult for your Council to carry on the work of the Society towards the end of the financial year, and to give the amount of matter in the *Journal* which it could and would give if the requisite funds were available. On the other hand, it will be noticed that, despite a slight increase in general expenditure, principally due to the reprinting of the rules, an allowance of £3 a month for clerical assistance, and depreciation on furniture, the total expenditure has been decreased by nearly £160.

The Balance Sheet shows that our immediate liabilities are exceeded by our cash assets by over £170, an eminently satisfactory position. The stock in hand of the Society's Transactions shows an increase in value of £276 1s. 7d., due to the cost upon completion of Volumes IV. and VII., and the addition of Volume VIII. The actual receipts from the sale of the several volumes amounted during the year to £218 18s. 8d., which amount has been applied to reducing the debit balances against each volume. Sundry debtors (of which £201 16s. 10d. is at our credit at Edinburgh, and is being drawn upon) show a decrease of nearly £100 compared with last year, and it is hoped during the coming year to eliminate this. There is a slight increase in the amount of provision for doubtful debts, due to the recovery of an outstanding account.

As usual, nothing has been taken into account for the unpaid subscriptions, amounting to £279 6s. for the past year. It is difficult to estimate how much of this is likely to be recovered but it may be mentioned here that the Council has, with great regret, been compelled to erase the names of 77 members, 31 associates, and 1 student from the roll for non-payment of arrears. The Council desires to point out that in no case has anyone been struck off, until he was over two years in arrears with his subscription.

Turning from the financial aspect to the practical work of the Society, it is pleasing to record that 62 members, 72 associates and 14 students have been elected and admitted during the year nine of the new members being associates who have transferred. The Council regrets that 42 members, 22 associates and 2 students have found it necessary to resign, while the Society has suffered the loss from death of the following 8 members and 4 associates :—

Members : Messrs. R. Camerer, E. D. Chester, T. Francis, C. Gluyas, F. G. Knox-Little, H. T. Pitt, W. Roff, and H. A. Tilghman.

Associates : Messrs. W. V. Hunt, C. L. Johnson, J. G. Johnson, and J. Perrin.

The tables with reference to yearly membership and its world-wide distribution are to be seen on the opposite page.

Twelve Ordinary General Meetings and one Special General Meeting have been held in the past year, at which the usual business of the Society was transacted, including the reading and discussion of many interesting and important papers.

The attendance at the monthly meetings has been maintained, the average attendance being 66.

Several interesting visits to mines have taken place during the year under review, and the educational and practical value of these excursions has been highly appreciated. These visits will be continued during the coming year.

The Annual Dinner was one of the most successful which the Society has yet held, the attendance being large and the speeches of considerable political and economic importance.

The *Journal* continues to be one of the best of its kind published in the mining world, and, according to reports received from various members while travelling in the United Kingdom, Australia, America, and elsewhere, its value to the mining and metallurgical profession is very highly spoken of.

The meetings of the Council have numbered 13, and the average attendance has been 13.

The attendances of the members of Council are as follows :—

Mr. F. F. Alexander	8	Dr. D. Macaulay ...	—
,, R. G. Bevington	10	,, J. Moir ...	10
,, W. A. Caldecott	4	Mr. S. H. Pearce ...	4
,, A. F. Crosse	1	,, A. Richardson...	9
,, W. R. Dowling	10	,, G. O. Smart ...	9
,, K. L. Graham...	7	Prof. G. H. Stanley	9
,, A. Heymann ...	10	Mr. A. Whitby ...	11
,, E. H. Johnson	4	,, H. A. White ...	13
,, A. McA. Johnston	10	Prof. J. A. Wilkinson	12
,, J. Littlejohn ...	6	,, J. Yates ...	12

Monthly meetings of the Publications and Finance Committees have taken place, at which the members of those committees have been most regular in their attendance. The Editorial Committee has also met when required, and the members of these committees have done a large

TABLE I.

	June, 1895.	June, 1896.	June, 1897.	June, 1898.	June, 1899.	June, 1902.	June, 1903.	June, 1904.	June, 1905.	June, 1906.	June, 1907.	June, 1908.
Hon. members ...	5	5	5	8	11	9	13	18	18	16	17	18
Life members ...	—	1	1	1	1	1	1	2	3	4	7	7
Members ...	49	81	101	197	281	291	441	639	680	680	683	618
Associates ...	—		—	180	238	243	281	338	321	314	342	357
Students ...	—	—	—	—			—	4	8	4	7	18
...												
	54	87	107	386	534	544	736	1,001	1,030	1,018	1,056	1,018

TABLE II.

DISTRIBUTION.	MEMBERS.			ASSOCIATES.	STUDENTS.	TOTAL.
	Honorary.	Life.	Ordinary			
Transvaal (Witwatersrand)	5	2	416	157	16	596
Do. (outside districts)	—	—	34	25	—	59
Rhodesia	—	—	53	42	—	95
Cape Colony, O.R.C., and Natal	1	1	6	3	—	11
British East & West, French W. & Portuguese E. Africa	—	—	4	3	—	7
British Isles	9	2	48	30	1	90
Australasia	—	1	7	22	—	30
Dominion of Canada ...	—	—	—	3	—	3
India and Burma	—	—	—	6	—	6
Europe	—	—	5	5	1	11
United States of America ...	3	1	21	28	—	53
Central and South America	—	—	5	1	—	6
Mexico	—	—	1	13	—	14
Borneo	—	—	—	2	—	2
Dutch East Indies ...	—	—	6	5	—	11
China and Korea	—		—	5	—	5
Federated Malay States ...	—	—	2	1	—	3
Addresses Wanted ..	—	—	10	6	—	16
TOTAL ...	18	7	618	357	18	1,018

amount of work, which the Society cannot too highly recognise.

At the beginning of the year your Council, in accordance with the resolution passed at the last Annual Meeting, transmitted to the Government a petition from the Society, praying that the *status quo* of the Patents Office be maintained. This petition, which was forwarded along with one from our sister society, the Transvaal Institute of Mechanical Engineers, did not, however, have the desired effect, but it brought from the Attorney-General a declaration, that the whole matter would be reconsidered later, possibly during the ensuing session.

The Seymour library still continues its career of usefulness, a large number of books having been added during the year, which will make its value much greater. In this connection our thanks are due to our representative on the Book Committee, Dr. Moir, and also to those members of the Society who have so willingly given their services voluntarily in sending in proposals for new books and journals. With the completion of the permanent building of the Transvaal University College, on Plein Square, the library will, during the year, be removed to special rooms provided for this purpose.

During the year many Commissions were appointed by the Government to investigate and report on matters connected with the gold industry. The Mining Regulations Commission, in dealing with the regulations obtaining on our mines, entered into a detailed and instructive investigation of ventilation, first aid work, sanitation, education, and the question of certificates, and much important evidence was laid before the Commission by members of the Society, including Prof. Yates, Dr. Moir, Dr. Irvine, and others. We were also well represented by our members before the Mining Industry Commission, which entered very largely into the labour question on these fields, and whose majority and minority reports are now before the public.

The President was appointed judge in the underground trials of the *South African Mines* Stope Drill Contest, recently held on the property of the Robinson G. M. Co., a competition which has resulted in a similar one being projected under the joint auspices of the Government and the Chamber of Mines, and to be known as the "Transvaal Stope Drill Competition, 1909." The President and many others of our members have been appointed on the Committee of these trials, and in many quarters it is expected that, as the result of the liberal prize scheme, amounting to £5,600, a satisfactory stope drill capable of largely supplanting hand drilling will be secured.

In view of the number of members outside South Africa the Council decided to recommend the appointment of representatives of the Society in the chief mining centres of the world. This recommendation was unanimously endorsed by a general meeting of members, and the election of suitable corresponding members of Council is now in progress. Besides marking the international scope of the Society's operations, it is hoped that this departure will serve to increase our membership, and to add increased interest and variety to our proceedings, by securing papers and contributions to discussion from the widest possible field.

Your Council is also co-operating with the Transvaal Institute of Mechanical Engineers and other institutions in the formation of a joint Standardisation Committee for the purpose of considering the advisability of altering the British Standard Specification for cement testing to suit local conditions. The question as to whether such committee might also deal with advantage with other materials affected by local conditions, such as local building material, timber, stone, etc., will also be considered.

In conclusion, the Society has once again to express its grateful thanks to the Transvaal Chamber of Mines for the use of the Council Chamber for our monthly meetings; to your Hon. Treasurer, Mr. James Littlejohn; to your Hon. Auditor, Mr. A. Aiken, for again auditing the Society's books and accounts; to your Hon. Solicitors, Messrs. van Hulsteyn, Feltham and Fry: and to the local Press for the space and publicity given to the proceedings of the Society, and notices of its various activities.

The President: I formally move the adoption of the report and accounts, and in doing so I should like to congratulate the members upon the strong position the Society is in to-day.

Mr. R. G. Bevington: I beg to second the resolution. In doing so I should like, as a member of the Finance Committee, to refer to the matter of the payment of subscriptions. As the Society has no revenue from invested funds, it is necessary that subscriptions should be paid at the commencement of the financial year in order to place the Council in the possession of money to work upon. I am sure it is only necessary to bring this matter to the notice of members in order to obtain their co-operation, and also to avoid the expense and trouble of many reminders. I also wish to impress upon members the fact that every consideration has been given before a member's name has been removed from the roll, more especially when a member has been in straitened circumstances. There are, however, quite a number of members who have not even replied once to the many notices issued, and there are also many who have gone away and left no address.

The Report and Accounts were adoped unanimously.

ELECTION OF OFFICERS.

The President then announced the election of officers, as contained in the following report from the scrutineers:—

Johannesburg, 20th June, 1908.

The Chairman,
Annual General Meeting of Members.

Dear Sir,—We hereby certify that we have this day examined 147 voting papers, handed to us by the Secretary, three of which were invalid for one reason or another.

The following is the result of our scrutiny:—

President:

Unanimously Elected. Mr. R. G. Bevington.

Vice-Presidents:

	Elected.	Votes.
1.	Mr. A. McA. Johnston	95
2.	Mr. T. Lane Carter	80
3.	Dr. J. Moir	70
	Not Elected.	
4.	Mr. W. R. Dowling	62
5.	Mr. H. A. White	58
6.	Mr. A. Whitby	46

Hon. Treasurer:

Unanimously Elected. Mr. James Littlejohn.

Members of Council:

	Elected.	
1.	Mr. W. R. Dowling	120
2.	Prof. G H. Stanley	119
3.	Mr. H. A. White	115
4.	Mr. A. Whitby	114
5.	Mr. K. L. Graham	112
6.	Prof J. A. Wilkinson	110
7.	Mr. C. B. Saner	101
8.	Mr. G. O. Smart	100
9.	Mr. A. Richardson	96
10.	Mr. C. B. Kingston	83
11.	Mr. J. E. Thomas	83
12.	Mr. E. H. Croghan	72

THE CHEMICAL, METALLURGICAL AND MINING SOCIETY OF SOUTH AFRICA.

(INCORPORATED MAY, 1907.)

REVENUE ACCOUNT for year ending 1st June, 1908.

Dr.

1906-7.			EXPENDITURE.	1907-8.					
£	s.	d.		£	s.	d.	£	s.	d.
(225	*5*	*7*	To CHARGES (including Postages, Attendance, Bank Charges, etc.) ...	292	9	4			
120	*7*	*2*	,, PRINTING AND STATIONERY ...	159	15	11			
107	*3*	*6*	,, RENT AND LIGHTING	90	12	0			
353	*0*	*0*	,, SALARIES...	386	0	0			
6	*16*	*4)*	,, DEPRECIATION ON FURNITURE ...	11	2	0			
							849	19	3
812	*12*	*7*							
			,, SOCIETY'S MEDALS				15	5	7
			,, SOCIETY'S JOURNAL—						
(1,162	*11*	*11*	Cost of Reporting, Printing and Sundries	918	6	11			
442	*18*	*0)*	*less* Received for Subscriptions and Advertisements ..	408	11	10			
719	*13*	*11*					509	15	1
109	*0*	*0*	,, BALANCE, surplus on year's working				233	4	5
£1,641	*6*	*6*					£1,608	4	4

Cr.

1906-7.			REVENUE.	1907-8.					
£	s.	d.		£	s.	d.	£	s.	d.
			By SUBSCRIPTIONS—						
(1,220	*16*	*0*	Members ..	1,174	6	0			
332	*17*	*0*	Associates	353	6	6			
2	*12*	*6)*	Students ..	6	16	6			
							1,534	9	0
1,556	*5*	*6*							
85	*1*	*0*	,, ENTRANCE FEES				59	17	0
			,, SURPLUS RECEIPTS OVER EXPENDITURE—Annual Dinner and Nigel Visit				13	18	4
£1,641	*6*	*6*					£1,608	4	4

BALANCE SHEET, as at 1st June, 1908.

1906-7.			LIABILITIES.	1907-8.					
£	s.	d.		£	s.	d.	£	s.	d.
(106	*10*	*8*	To SUNDRY CREDITORS	3	16	2			
30	*19*	*6*	,, SUBSCRIPTIONS paid in advance ...	53	11	0			
35	*0*	*0)*	,, PRIZE FUND	27	0	0			
							84	7	2
172	*10*	*2*							
			,, REVENUE ACCOUNT—						
(889	*2*	*10*	Balance at May 31, 1907 ...	998	2	10			
109	*0*	*0)*	Add surplus as per Revenue Account, June 1, 1907 ...	233	4	5			
							1,231	7	3
998	*2*	*10*							
£1,170	*13*	*0*					£1,315	14	5

1906-7.			ASSETS.	1907-8.					
£	s.	d.		£	s.	d.	£	s.	d.
(231	*15*	*4*	By CASH AT BANK	203	15	6			
	15	*3)*	,, CASH IN HAND	4	11	0			
							208	6	6
232	*10*	*7*							
20	*0*	*0*	,, OFFICE FURNITURE				15	0	0
(110	*10*	*4*	,, TRANSACTIONS, VOL. II.—Stock in hand	99	19	0			
82	*5*	*3*	,, ,, VOL. III. ,,	32	8	8			
			,, ,, VOL. IV. ..	226	10	0			
128	*2*	*8*	,, JOURNAL, Vol. V., Stock in hand ..	119	3	5			
154	*11*	*0*	,, ,, Vol. VI.	138	11	9			
92	*17*	*0)*	,, ,, Vol. VII. ,, ..	145	15	0			
			,, Vol. VIII. ,, ...	82	0	0			
568	*6*	*3*					844	7	10
(358	*13*	*2*	,, SUNDRY DEBTORS	263	16	7			
8	*17*	*0)*	*less* Provision for Doubtful Debts	15	16	6			
							248	0	1
349	*16*	*2*							
£1,170	*13*	*0*					£1,315	14	5

I have examined the foregoing Statements of Account, compared them with the Books and Vouchers of the Society, and I find the statements to be true and correct, as shown thereby.

Sgd. ALEX. AIKEN, *Incorporated Accountant,* } *Hon. Auditor.*

JOHANNESBURG, JUNE, 1908.

(Signed) J. LITTLEJOHN, *Hon. Treasurer.*

(Signed) FRED. ROWLAND, *Secretary,*
Fellow Chartered Inst. Secretaries.

Not Elected.

13.	Mr. A. Heymann	52
14.	Mr. G. Melvill	46
15.	Mr. W. Beaver	42
16.	Mr. J. T. Milligan	39
17.	Mr. W. Taylor	34
18.	Mr. J. P. Ward	29

Yours faithfully,
Sgd. JAS. GRAY,
,, JAS. LEA,
,, P. T. MORRISBY, *Scrutineers.*
,, A. L. EDWARDS,
,, W. E. THORPE,

The President: I think we are to be congratulated on the new officers. I really do think we have done remarkably well in our selection, and on your behalf I extend a hearty welcome to the new President and the other members of the Council.

Mr. R. G. Bevington: I am deeply sensible of the great honour which has fallen upon me in becoming President of this Society. I hope, with your co-operation, I shall be able to go through with it, and hand the office over to my successor as Prof. Yates does to me. I trust that during the ensuing year you will assist me in every possible way, so that our Society may continue to be the success it has been in the past, and, if possible, to make it a greater success.

The President: In connection with the elections, we would be lacking in our duty if we did not propose a vote of thanks to the scrutineers who have devoted most of the day to the ballot. I move a vote of thanks to those gentlemen for their work.

Mr. Jas. Gray: On behalf of the other scrutineers and myself I would state that it was a pleasure for us to do the work, and we desire to record our thanks for the hospitality extended by the Society through the Secretary.

AUDITOR.

Mr. Aiken was appointed auditor on the motion of Mr. W. Cullen, seconded by Prof. J. A. Wilkinson.

PRESENTATION OF MEDALS.

The President: It is nearly two years since Drs. Macaulay and Irvine gave us their paper on "Safety Measures in Mining," the discussion on which you remember continued about twelve months, in fact, so long that it tided over the last annual meeting, and therefore it was not possible to present the medals at that meeting. Then Dr. Irvine went home for six months, and on his return Dr. Macaulay left us for four months. This explains the somewhat belated presentation, which is about to be made. It is a very great pleasure for me to-night to present these medals to Drs. Irvine and Macaulay, and I think they are well entitled to this reward. It is, as I have said, sometime since this room witnessed that notable discussion, but although the paper has become a matter of history, yet every one who goes along the reef sees daily the result of that paper and the discussion thereon. We see it in the improved conditions which are obtaining at present. It is indeed a great work this looking after the health of our miners, and our friends Drs. Macaulay and Irvine are well entitled to all that we have given them. I have also to refer to Mr. Whitby, who, I am sorry to say, has just been undergoing an operation, but is progressing very favourably. He is one of the hardest workers on our Council, and the diploma which he is about to receive is a small token of our recognition of his work and of his paper "Routine Assaying on the Rand." We all hope he will have a speedy recovery and that he will come along as frequently in the future as in the past. I have the honour and pleasure of presenting you, Dr. Macaulay and Dr. Irvine, with these medals and diplomas, and we express the hope that you will long continue in the good work that you have so well begun.

Dr. D. Macaulay, M.L.A.: We have talked so much and to such good effect in this Society, that is unnecessary for me to say very much on this occasion, except to thank you for this appreciation of the little work which we were enabled and privileged to do for the Society, and also for the mining industry in this country. If I were going to dilate on this topic I might say that this might be reckoned a memorable occasion in the history of your Society as well as in the history of the mining industry of this country; that is to say, you have found occasion to present your highest award to two humble workers in a field which is only apparently remotely associated with the work which you are engaged upon, but which you found, when brought to your notice, was intimately associated with the welfare and condition of work of the men on these fields. Dr. Irvine and myself have found our chief satisfaction in the fact that the work which we have done with your assistance and the criticisms which you were able to make has brought to the notice of the powers that be a very important aspect of mining, and we hope that the result will be to prove what we asserted in our second paper, namely, that mining in this country can be made as healthy as any other kind of work.

Dr. L. G. Irvine: I have little to add to what Dr. Macaulay has said. We both thank you very warmly for this mark of your appreciation. I think it is a very great tribute to the catholicity which animates your work in this Society that you, as Dr. Macaulay has said, have given these diplomas and medals to the writers of a paper which was quite non-technical, which was really more medical than technical, but which, nevertheless, you recognised as being intimately associated with the work which you are carrying on. Since we came into this Society we have always recognised this catholic spirit which you especially showed by the welcome you gave us when we brought forward that particular aspect of mining work. Reform comes very slowly. It is a very long time since we discussed these things, and there is still a good deal to be done, but I hope, that as a result of the work of the Commission which is sitting now, we shall get a step or two further in advance. Dr. Macaulay and I want to see the leaders of the mining industry and those associated with the work of the mines taking these matters to heart. We want to see the initiative in health reform come, not from the Mines Department, but from the mining industry and from the mine officials themselves. Because we are quite convinced that it only wants really serious and earnest co-operation between mining officials from the highest to the lowest to make an immense improvement in the conditions which now obtain. That would be to us the greatest reward for the little work we have done, to see the people connected with mining take these matters to heart, and do them off their own bat, and not be compelled to give effect to the measures which are so necessary to make the conditions of mining here as perfect as they can be.

The President: In connection with these medals, I move a vote of thanks to Dr. Kirke Rose of the Royal Mint, London, who rendered us such valuable assistance in connection with the procuring of these medals.

This motion was carried by acclamation.

General Business.

Mr. E. J. Laschinger: Under the heading of general business there is a matter which I should like to bring forward, and which I have very much at heart, not only as a member of this Society, but as a member of other technical societies on the Rand. The opening for the matter which I am about to bring forward was very aptly given me by the new President, who referred to co-operation. It is on occasions of this kind that matters of policy are properly discussed, and at the last annual meeting quite a lot of matters of policy were brought forward. What I want to see is more co-operation between the technical societies of the Rand. I wish no one to be under any misapprehension as to what I mean. I do not mean amalgamation or affiliation, or anything of that sort. But there are various matters on which the technical men of the Rand can co-operate, because they are equally interested. One aspect is the social aspect. We have here quite a number of technical societies, of which this is numerically the strongest, and I, therefore, wish to bring this matter before this Society especially.

There are more technical men concentrated in a small area here than in any other centre in the world, and we do not come enough into contact with each other. There is too much separatism. We ought to know each other better, and the only way to do that is to have some kind of social intercourse in rooms set apart for the purpose, rooms open all day and into the evening, with some one to look after them. We would have a library there, and men from the mines all along the Reef could at any moment look at the exchanges of all the technical journals of the world. If a suggestion emanated from the Chemical, Metallurgical and Mining Society, I believe it would be warmly welcomed by other kindred bodies, and I hope that this year will see inaugurated more of the co-operative spirit. In conclusion, I should like to refer to the very fine report which the Council have presented to-night. There is nothing in the report which indicates that the Council have got "swelled head," and there is nothing there upon which the Council congratulate themselves too highly or inflate themselves with pride, and it is quite possible that they had in mind one of the sayings of the Old Book, "Pride goeth before destruction, and a haughty spirit before a fall."

Prof. J. A. Wilkinson: Has Mr. Laschinger any motion to bring forward on this matter, because we have heard of this before in the Chemical Society, and I must say, I am surprised that this stalking horse has been led in again by Mr. Laschinger.

The President: Possibly you regard this as too serious a matter to be discussed to-night. The new Council will bear it in mind.

Mr. E. J. Laschinger: With regard to the question raised by Prof. Wilkinson, I have no hesitation in putting it to the meeting as a simple proposition. I should like to put it forward in the positive sense, "That it is the feeling of this meeting that a greater amount of co operation amongst the various technical societies on the Rand will be to the benefit of the technical professions here."

The resolution was duly seconded.

Prof. J. A. Wilkinson: Is this motion in order?

The President: I believe it is quite in order.

Mr. C. T. Gardiner: I think there should be more notice given of a resolution like this. I am rather inclined, with all deference to Mr. Laschinger, to think that this approaches amalgamation. There are difficulties which I will not dwell on now, and I think this resolution should be brought up at another meeting, after due notice, and then the feeling of the meeting can be tested.

The President: I would like to point out that the resolution really commits the Society to nothing, and it is therefore for you now to either move an amendment or accept the original motion.

Prof. J. A. Wilkinson: I do not wish to raise an obstruction to the spirit of the motion which Mr. Laschinger has brought forward, but, as I said, this is not the first time such a proposition has been discussed here. If it has not been discussed in open meeting, it certainly has been discussed for some long time past in Council. This matter of co-operation is purely a matter of finance. Now, we are not financially strong. Every member of the Finance Committee knows that we have had to struggle hard to present the balance sheet, which has been placed before you to-night. You all know this is a time of depression, and, therefore, I do not think any member of the Council can look forward to the coming year with any sense of assurance in his mind, that at the end of the year we are going to come out square. There is another point. If you pass this motion this evening, you practically tell us that there has been a lack of co-operation in the past amongst the scientific societies. Now, I for one, deny that. When any motion has come forward from any one of the other societies, it has been accepted and discussed by your Council in the most hearty and unanimous spirit. A recent example of that has been the request of the Transvaal Institute of Mechanical Engineers to join with them in altering or, at any rate, revising the British standard specification for cement, a request with which we readily complied. One other fact which seems to have been forgotten I wish to mention. You all know the new building of the Transvaal University College, which is now approaching completion, and all of you, who have taken any interest in that scheme, know that it was in the minds of those who promoted it, that part of that building should be used to house the scientific societies. There are certain rooms in that building which have been set apart to house scientific societies. I do not know what it is that Mr. Laschinger wants, but a motion such as this in such general terms does not appeal to me. I am with the spirit of it, and, as one of the Council, I will do all in my power to further this spirit of co-operation. I ask you not to pass this resolution now, because I feel with the last speaker, that it would be an instruction to the Council to do something which they are assumed not to have done in the past.

Mr. C. T. Gardiner: I move that this matter be discussed six months hence.

Prof. J. A. Wilkinson seconded.

Mr. E. J. Laschinger: I do not wish it to be thought that I am bringing this forward as something new. I am well aware of the facts which Prof. Wilkinson has brought forward about these rooms, but you will see at once that even if these rooms are set apart, unless the societies co-operate there is going to be a discussion as to who is going to have the best room and so on. It is simply a matter, which I thought should be brought up at a meeting of this kind. There are so many ways in which co-operation can be to the advantage of each society in lessening expenditure instead of increasing it. The object of the motion is simply to test the feeling of the members.

Mr. A. McA. Johnston: I would put it to Mr. Laschinger whether he is willing to let it go forward to a future meeting of the Council, and ask them to bring it forward to a general meeting.

Prof. J. A. Wilkinson: I would be in favour of that.

Mr. E. J. Laschinger intimated that he wished his resolution to go to vote.

The President then put the amendment, which was lost, and declared the motion passed.

I wish to remind you that Mr. Lane Carter, one of our esteemed Vice-Presidents, is going home for a well earned holiday. He has the good fortune to be the manager of the French Rand G. M. Co. This mine is not exactly a Bonanza, but it is one of those mines where one can obtain a heap of experience, hence Mr. Carter's good fortune. We shall miss him very much, but in the meantime we wish him a very pleasant holiday and a safe return.

Mr. T. Lane Carter: I thank you very much for your kind expressions. If anyone wants experience, the French Rand is the place to go and get it. I can assure you I will not forget this Society in the many wanderings I expect to take during the next six months. I shall keep my eyes open, and if there is anything I see that will interest you, I hope that in the intervals of travelling about from place to place, I shall be able to jot down a few notes, and send them forward. I also hope to take part in the discussions and, generally, to keep in touch with you as much as possible. I will not promise to do what our friend, Mr. Caldecott, did when he returned, namely, give you a paper, but, at any rate, I hope I shall come back full of ideas, and it will always be my pleasure to give them to this Society.

The President: It is not my intention to give you a valedictory address, for I feel sure that the review of the past year's work can very well be left in the hands of Mr. Bevington, who is a well tried member of the Council, and who for many years has been doing very valuable work on behalf of the Society. I feel sure that during the coming year he will maintain the high standing of the Society.

The next item on the agenda is the paper by the late Mr. Harry Pitt, which Mr. Bevington will read.

NOTES ON THE STAMP MILL WATER FEED AND PACKED UP DIES, INTRODUCING THE SHALLOW FRONT MORTAR BOX.

By the late HARRY T. PITT (Member.)

A great deal has been said lately for, and (by some who have not tried it) against, the back water feed for stamp batteries, and the packing up of dies, so as to maintain a level discharge and even crushing throughout the life of the dies.

I am, therefore, inclined to think that the following figures (though somewhat incomplete through loss of data during the war period), will interest members of this Society.

They are the results of trials made by me at the Rose Deep mill in August, 1899: both with the back water feed and with packed up dies. To trot out figures collected in 1899, savours of ancient history; as a matter of fact, we are pre-war with our mortar boxes, if not prehistoric. At any rate, little has been done to strengthen them or to shape them so as to help up the number of tons crushed per stamp. Practical millmen will, I am sure, admit that there is room for improvement.

The old method of introducing the water at the top of the mortar box, delivers the water on top of the stamp heads. It then falls on to the dies like a rain shower, and consequently does not assist in forcing the crushed pulp to the screens. When water is fed through the ore feed chutes, that which does not strike the heads or shoes, shoots over the dies and lands on the bottom of the screen frames or chuck blocks, thereby causing a back-wash which has a tendency to keep back crushing, by checking the free discharge.

Water let in at the back of the box, level with the top of the dies, as in trial No. VI., forces the uncrushed rock to bank up against the screens and causes the shoes to drop on bare dies; further, the water inlets will frequently become choked with small rock. If water be let in through five pipes, two at each side of the feed chute, and one pipe under the feed chute, so as to deliver on to the back plate vertically and strike the back end of the dies at an angle of about 50°, as shown in sketch, then the water striking the back plate, comes off on to the dies in a fanlike stream, not strong enough to force the uncrushed rock to the screens, while having sufficient force to carry all crushed rock and slimes to the screens, to be helped out with the splash caused by the dropping of the stamps. This water striking the dies at an angle of about 50° will follow the die down as it wears. This fact is of considerable advantage, especially where deep mortar boxes are used.

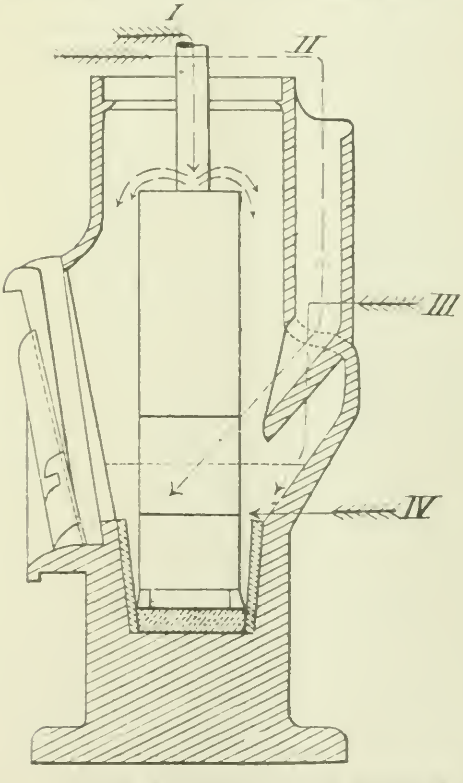

FIG. 1.—SKETCH SHOWING VARIOUS METHODS OF SUPPLYING WATER.

I advocate the style of water feed as in trial No. III., not only from results obtained through experiments, but from close observation of operation on a large scale.

By packing up the dies with false bottoms and using different sized chuck blocks (as the dies wear down) an even crushing throughout the life of the dies can be maintained. Another method to attain the same end is to use Mr. Chew's false dies. But to change false dies and false bottoms takes up time and causes stoppages and is costly in spares. I take it that all millmen are endeavouring to crush a large quantity of rock, and to crush it cheaply in the shortest space of time. I make the above fact my excuse for introducing the following sketch of a "shallow front mortar box." With such a box the dies can be worn down to the flange which holds them in position and the discharge maintained evenly all the way down by simply changing the chuck block as the dies wear.

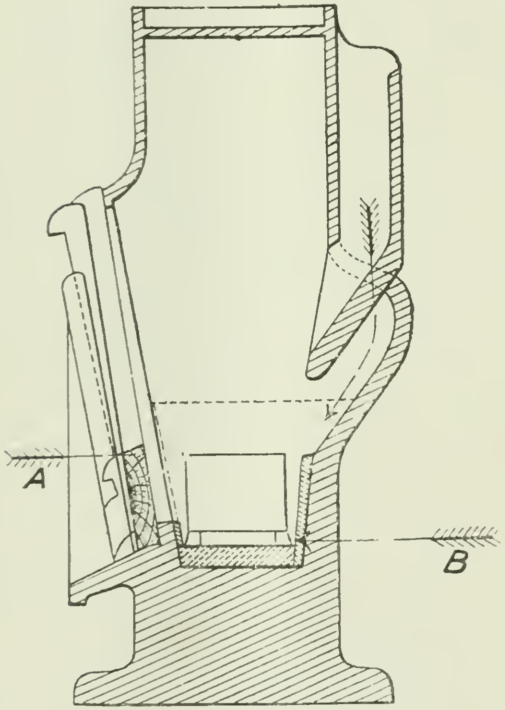

FIG. II.—SKETCH OF BOX.

A shallow front mortar box will not cost any more to construct than those at present in use, and no extra spares are required. The chuck blocks can be changed at any convenient time as easily as a screen is changed. For this box I would advocate the use of a 15-in. screen instead of 12 in., so as to take full advantage of the splash. This box will cost less in liners, as will be seen on reference to the sketch, and there will be less iron or steel used in casting them.

Water let into box, as shown by arrows, will follow dies down to the bottom of the box. With liners made as per sketch, it will not be necessary to take out false bottoms when changing liners.

By decreasing the height of chuck blocks A as the die wears, a level discharge can be maintained until the die is worn down to the flange B (which holds the die in position).

Unfortunately, I was not allowed to spend any money in partitioning off the mill bins and equipping boxes so as to carry on my experiments with back water feed and raised dies. However, Mr. Caldecott, of the Consolidated Goldfields, took the matter up and carried out trials at the Knights Deep mill with and without the back water feed, and he can, I am sure, give some very interesting figures on results obtained. There are several mills on the Rand now equipped with back water feed, amongst them being, the Knights Deep, Rose Deep, Glen Deep and others. Of this, I am certain, the duty per stamp is increased thereby, and the cost of installation is no more than the ordinary feed. In conclusion, I would say, if this little paper only succeeds in drawing milling and other members of the Society into a discussion on the water feeds in use in the many different mills on the Rand, so as to bring us to some definite decision as to which is really the most beneficial water feed to use, it will have served its purpose.

TRIAL NO. I.—*Working Conditions.*

Weight of stamp	1,150 lb.
Height of drop	7 in.
Number of drops per minute ...	105
Size of screening	500 light mesh
Screen area	520 sq. in.

Water supply let into top of mortar box.
Discharge level with the face of dies.
Dies packed up on two false bottoms.
Chuck blocks decreasing in height as dies wear.

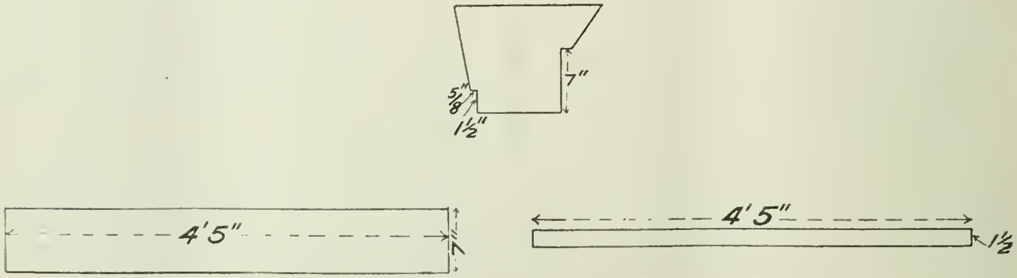

FIG. III.—SKETCH OF FRONT, BACK AND SIDE LINERS.

Results.—Course Rock.—Five stamps took 39 hours 47 minutes to crush 51 tons of 2,000 lb. = 6·153 tons per stamp per 24 hours net.

Screened Rock (passed through 1½ in.).—Five stamps took 27 hours 38 minutes to crush 36 tons of 2,000 lb. = 6·253 tons per stamp per 24 hours net.

Trial No. II.

All conditions exactly the same except the discharge was not level, but 1 in. above the die level.

Results.—Course Rock.—Five stamps took 15 hours 56 minutes to crush 19 tons of 2,000 lb. = 5·723 tons per stamp per 24 hours net.

Screened Rock (1½ in. mesh).—Five stamps took 35 hours 36 minutes to crush 43 tons of 2,000 lb. = 5·797 tons per stamp per 24 hours net.

Trial No. III.

All conditions exactly the same as in trial No. II. except the water supply, which was let in through four 1½ in. holes, level with the ore feed chute at the back of mortar box. Dies 1 in. below discharge level. Two false bottoms.

Results.—Course Rock.—Five stamps took 9 hours 12 minutes to crush 11 tons of 2,000 lb. = 5·740 tons per stamp per 24 hours net.

Screened Rock (all fines).—Five stamps took 11 hours 5 minutes to crush 20·5 tons of 2,000 lb. = 8·878 tons per stamp per 24 hours net.

Trial No. IV.

All conditions exactly the same as in trial No. III. except that the discharge was level instead of being 1 in. above the die level.

Results.—Course Rock.—Five stamps took 190 hours 4 minutes to crush 236 tons of 2,000 lb. = 5·941 tons per stamp per 24 hours net.

Screened Rock (1½ in. mesh).—Five stamps took 84 hours 43 minutes to crush 106·5 tons of 2,000 lb. = 6·034 tons per stamp per 24 hours net.

Trial No. V.

All conditions exactly the same as in trial No. IV. except the screen, which instead of being straight was made to give 173 sq. in. more area than the straight screen.

Results.—Course Rock.—Five stamps took 33 hours 16 minutes to crush 400 tons of 2,000 lb. = 5·771 tons per stamp per 24 hours net.

Screened Rock (1½ in. mesh).—Five stamps took 73 hours 13 minutes to crush 97·5 tons of 2,000 lb. = 6·390 tons per stamp per 24 hours net

Trial No. VI.—Battery No. 20.

All conditions exactly the same as in trial No. I. except the water supply, which was led in through five 1¼ in. holes in back of mortar boxes, level with the top of dies and discharge on box lip. Two false bottoms.

Results.—Course Rock.—Five stamps took 191 hours 29 minutes to crush 261·5 tons of 2,000 lb. = 6·630 tons per stamp per 24 hours.

Screened Rock (1½ in. mesh).—Five stamps took 324 hours 51 minutes to crush 418·5 tons of 2,000 lb. = 6·627 tons per stamp per 24 hours net.

The President: There is a melancholy interest attached to this paper, because of the death of Mr. Pitt. The Railway Pioneer Regiment, which was raised from among the employees of Rand Mining Companies, largely through the efforts of the late Major Seymour, you will all remember well. You will also remember the disastrous day at Sand River, in the O.R.C., when Major Seymour, one of the finest engineers that the Rand has ever seen, Lieutenant Clement and others, were shot. It was in the same engagement that Mr. Pitt received the wound which recently necessitated an operation, and unfortunately that operation has ended fatally. I move a vote of thanks to Mr. Campbell Pitt, his brother, for sending forward these notes and allowing them to be read.

Mr. R. G. Bevington: I will second the vote of thanks. Mr. Pitt has placed before us some very interesting notes, and I know it was his intention, had he been spared, to have pursued those investigations further. So that we have not only to deplore the loss of a valuable member of our Society and a man whose heart was in his work, but also the fact that we are denied a further contribution from him on this interesting subject. As there can be no reply, I do not see how we can discuss this paper in the ordinary way, but there are one or two points which strike me and which I should like to allude to. Firstly, you will see from the sketch that his experiments were carried out with the pattern of mortar box which has a recess at the back for the purpose of holding a copper plate, when such is used, with inside amalgamation, and I gather that it was Mr. Pitt's plan to admit the water by pipes fitted through the roof of this recess. Now as I do not use inside amalgamation, I have, as many others have also done, filled up this recess with a block of wood faced with an iron plate so as to convert the boxes, as nearly as possible, to the straight back pattern. In this case, in order to obtain as closely as possible, the results of the method of application of water, as favoured by Mr. Pitt, it has always been my practice to admit the water by means of two 1 in. pipes at the top of the box between stamps 1—2 and 4—5 so placed as to direct the stream of water against the back of the box over which it spreads itself and runs down gently in a sheet, tending to wash the crushed rock and slime from the back of the box towards the screens. The shallow front box seems quite a good idea and well worth trying. Perhaps some one of the manufacturers of these articles will be prepared to make one and get it tried as an experiment. Turning to the table of trials, you will notice a very important point when trial No. III. is compared with the others. This trial is the only one in which the stamps were fed with "all fines": in the other trials the finest rock fed having been fed through 1½ in. mesh sieve. Now the increase in the output in crushing the "all fines" as against the other trials is so enormous as to, I think, dispose of the theory so often advanced that it is no good crushing rock too fine before sending it to the stamps.

Mr. W. A. Caldecott (contributed): The discussion on the paper written by the late Mr. H. T. Pitt affords a welcome opportunity of paying a tribute to his memory, as one of the pioneers upon these fields, and even earlier at Barberton. His sterling work, both in the field of ore treatment and during the war, is known to us all, and in losing him we have lost a typical representative of those members who constitute, by their steady devotion to duty and to the companies they serve, the true backbone and strength of our Society.

As mentioned by Mr. Pitt, I adopted, upon his suggestion, the method of back water feed in some experimental work proceeding some years ago at the Knights Deep, and the suggestion has since been acknowledged.* Besides serving as a good method of introducing water into the mortar boxes under any circumstances, this feature is especially valuable, when fine crushing with high discharge, and it has been incorporated in the structure of over a hundred mortar boxes cast for the Consolidated Gold Fields, since the experiments above referred to. At the Homestake, crushing with a 10 in. height of discharge, some ten or twelve tons of water per ton of ore are required with the ordinary method of supplying water, for creating sufficient agitation to wash out the particles of ore. At the Simmer East we are able, when fine-crushing in the battery with the same height of discharge as above, to employ the ordinary water ratios (say, seven to one), because the gently uprising current of water introduced near the die level serves to carry upwards, by a process of elutriation, particles fine enough to pass the screen. The importance of a not excessive water ratio becomes evident when it is considered that upon the volume of pulp depends the plate area, cost of elevating pulp, number of classifiers, launder and collecting vat capacity, and cost of pumping water back to the mill.

A LABORATORY COMPARISON OF TUBE MILL PEBBLES.

By Prof. G. H. Stanley, A.R.S.M., M.I.M.E., M.I.M.M. (Member).

With an Appendix on Liners, by M. Weber.

The following brief account of a comparative examination of samples of local and imported pebbles, will, I am sure, be of interest to those members of the Society who are concerned in running tube mills and, doubtless, to others as well, even though the increasing use of banket lumps as a substitute renders the subject of pebbles less important, at any rate in Rand gold milling practice.

The local pebbles examined were rather smaller than the imported, and of more irregular shape, their colour being grey, with white and yellowish white patches, that of the imported being greyish white. The local pebbles were more easily broken under the hammer, and considerably easier to grind on the bucking plate; their interior colours varied from light brownish grey to nearly black in different pebbles, and their fractures, although conchoidal like the imported, and fine grained, were obviously to the naked eye much coarser in grain than the latter, the fracture of which was of the same colour as the interior, and very fine grained. Very little difference in hardness was noted by the test of relative ability to scratch one another, but it was rather easier to scratch local by imported pebbles than *vice versa*, and this was confirmed by the amounts of iron, removed from the bucking plate during grinding and taken up by the sample, subsequently removed by a magnet. The local pebbles took up about 2 %, and the imported considerably more.

In order to ascertain whether chemical composition could in any way account for these observed differences, fair samples of each type of pebble were taken and analysed with the following results:—

	Local Pebbles. %	Imported Pebbles. %
Loss on ignition (H_2O, CO_2, etc.)	0·46	1·00
Silica (SiO_2)	97·20	96·78
Iron oxide and alumina (Fe_2C_3 and Al_2O_3)... ...	1·95	1·86
Lime (CaO)	0·22	0·26
Magnesia (MgO)	0·16	0·07

The composition of the two classes, therefore, was practically identical, and furnished no clue to the different properties or behaviour. As already stated, the local pebbles were evidently of comparatively coarse structure, so that the next step

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*See *Proceedings of the Institution of Mining and Metallurgy*, vol. xiv., 1904-5, p. 164.

was to have thin sections prepared for microscopic examination in order that the internal structure could be studied, and this, as shown in the photographs, furnishes, I think, the explanation sought for.

Both photographs were taken with polarised light and show a magnification of 75 diameters. In No. I., a local pebble, it is seen that the section is not only composed of comparatively coarse grains with nearly smooth edges (as distinct from photo III.), but that the grains are not of uniform size, in fact, the mass is composed of mosaic secondary quartz with a few specks of other minerals, and is really a quartzite. The grains do not appear to be cemented together, but are held together, though not strongly, mainly by interlocking, thus creating difficulty in preparing the slide as they easily separated during grinding. This accounted for the comparative brittleness of these pebbles, as the difference in sizes of grains may account for uneven wearing.

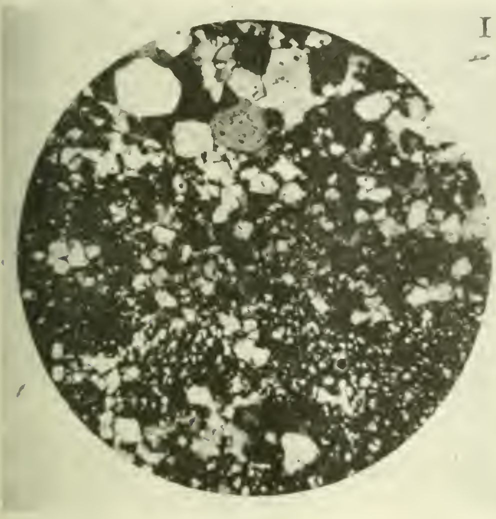
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FIG. I.—LOCAL PEBBLE × 75.

In No. II., on the other hand, an imported pebble, the structure is crypto-crystalline and homogeneous, and in places distinctly radiating. It is probably composed therefore of the Chalcedonic variety of silica, *i.e.*, a true flint, and no difficulty was met with in making the slide. This greater brittleness and slightly inferior hardness of the local pebbles, of course, influences their wearing qualities, the imported pebbles lasting longer, though the difference, I believe, is not very considerable and may be offset by the lower cost. It is, however, obvious, that considering only the local pebbles, the more the quartz grains interlock the less easily will they be separated and the less brittle will the pebble be. In some quartzites this interlocking is very marked, and the rock is in consequence comparatively tough and strong, and it would seem, therefore, that the microscope may furnish valuable information as to the suitability of proposed pebbles, and also liners, for employment in tube mills.

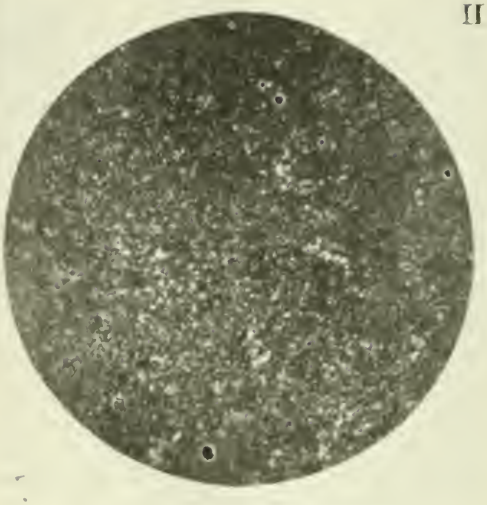
II

FIG. II.—IMPORTED PEBBLE × 75.

With regard to liners, Mr. Weber has kindly furnished me with the appended note embodying some of his observations, and, in conclusion, I may perhaps offer an opinion, that in tube mill pebbles and liners the important properties are three in number ; hardness, toughness, and homogeneity, the order of importance in the case of pebbles being as given, and in the case of liners, hardness, homogeneity and toughness.

Hardness is a *sine quâ non* : homogeneity results in even wear, of great importance with liners ; while pebbles particularly require toughness to resist breaking when tumbling about in the mill. The imported pebbles and liners apparently excel in each of these qualities.

Liners (by M. Weber).—During the last twelve months, I had on several occasions to study and compare rocks which, either had been used, or were to be used in mining tube mills. I can only confirm what Prof. Stanley says with regard to the structure and its bearing on the tenacity of the rock. I have seen some local chert sections which had a fine and uniform grain, but I never found a piece of more than 2 in. in diameter in which there were no zones of coarse crystallisation, like those shown in photograph No. I. of a local pebble.

I was recently asked to determine which of two local specimens would probably last longer as a tube mill liner. The one I selected was to all outward appearance a hard glassy quartzite, which the microscope, however, showed had originally been a sandstone. The original quartz grains are best seen in ordinary light. They are small, well rounded, and comparatively far apart. The general appearance of the rock, under magnification, is that of a somewhat gritty sandstone. The interstices between the quartz grains are filled with clear quartz, and the outlines of the quartz grains are marked by dustlike opaque inclusions. Polarised light reveals the fact that the interstitial filling, which usually in a quartzite consists of a mosaic of optically differently arranged minute secondary quartz grains, disappears, *i.e*, the original round quartz grains have grown at the expense of the secondary cement, which ultimately has been altogether replaced by extended portions of the quartz grains, each individual grain having grown until it was met and stopped by the extending portion of the next grain. In polarised light, therefore, no cement is seen, but the whole of the rock consists of angular elongated crystals of quartz dovetailed and interlocked tightly together on account of their serrated edges, and inside which one still recognises the outline of the impure original quartz grain. This structure is shown in photograph No. III., and it is to this that I attribute largely the exceptional wearing quality of the rock. During examination uniform wavy extinction, due to molecular rearrangement produced by pressure was noticed and suggested that the hardness of the rock might have altered. On polishing a face I actually found that an ordinary quartz crystal did not make any incision into the polished face, *i.e.*, the hardness was more than seven, and obviously increases resistance to wear. A lining of this material was put in and has been used up to the present for 85 days, behaving quite satisfactorily and appearing likely to last 120.

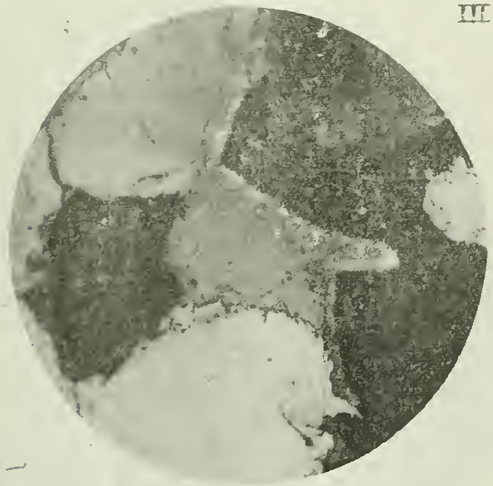

FIG. III.—LOCAL LINER ×85.

The other specimen was a dark coloured chert with white rims surrounding crystalline quartz. Applying the same test to a polished piece I found that the black portions were easily scratched by quartz. The white rims surrounding the crystalline quartz were found to be harder than quartz, so that the face of this rock in use as a liner will necessarily, after a short time, become uneven and the harder portions will obviously break off when the softer material around them is worn away. The microscopic characters were similar to those shown in photo I.

There is still one more point in favour of the quartzite as a liner. The rock has to be dressed to a certain shape, and a rock which has in all directions the same hardness and tenacity lends itself ever so much better to this operation than the chert.

The President moved a vote of thanks to Prof. Stanley for his interesting discussion of the important subject of tube mill pebbles. This was agreed to.

ASSAY OF BATTERY CHIPS AND SCREENS.

(*Read at February Meeting, 1908.*)

By Leslie J. Wilmoth (Associate).

DISCUSSION.

Mr. H. R. Jolly: In answer to Mr. White's question regarding the addition of sodium sulphate direct instead of sodium bisulphate, I wish to say that sodium sulphate was tried repeatedly, but it did not have the effect on the fluxing of the large pieces of iron that sodium bisulphate had. There was no apparent evolution of gas in the earlier stages of fusion, such as might arise from interaction between the free acid of the bisulphate and the soda, and, in fact, the fusions took place in a very ordinary manner. That the bisulphate exerted some powerful action on the fragments of iron is evident, as no attempts to slag those without it were satisfactory. The method appears to Mr. White round about and unscientific, but I fail to see how he arrives at that conclusion. He evidently has overlooked the remarks I made at the beginning of my paper, when I said that it was not my intention to theorise, but to give a few notes embodying the practical results of some experiments. My critic has seized a well-known wet reaction, applied it to bodies undergoing fusion, and, in fact, theorises without practice. No doubt, if he will take the trouble to make a few experiments on the flux suggested by me in a practical manner, he may see reason to modify his views.

Mr. H. A. White: It is rather late to again intervene in this discussion, but I cannot allow a question like this to go against me by default. It must be clear to most of us that the reaction between bisulphate and carbonate of soda is not only a wet one, but also obtains at the heat of dry fusion. It follows that Mr. Jolly's method amounts to the addition of sodium sulphate, and no amount of practical tests showing a certain difference in effect under special and varying circumstances can get away from this difficulty. However, my chief reason for again emphasising this point is that I deplore the introduction of fresh empiric methods into assaying more than is necessary. Even at the present time the process is not so scientific as most wet analytical methods used commercially and I, therefore, think that when any new practical method like this is introduced, the theoretical basis of it should not be ignored. In the original paper the new method there introduced is well supported by consideration of reactions involved as well as by experimental demonstration of the results obtained.

REPLY TO DISCUSSION.

Mr. L. J. Wilmoth: In reply to the query of Mr. McKeown, I should think that the storage of old battery screens would affect their value owing to dilution by the oxygen taken up during oxidation. Mr. Jolly evidently finds the matte assay a little difficult, but I think a little more perseverance would have overcome his difficulties. A button, such as he obtained, certainly points to a serious defect in his assays. It is evident that he saw the weak point from the remark "that there must have been a large quantity of iron or iron sulphide carried down with the lead." It is unfortunate that he did not attempt to get over this difficulty, which is due not to any defect in the method but to a defect in its application. It would seem that the trouble lies with the metallic iron not being completely taken up into the matte. It is probable that if he had added a little more sulphur and kept his pots in the fire a little longer his difficulties would have disappeared. With sufficient heat I fail to see how the matte could be mixed with the lead button. The high values obtained in the matte are hardly surprising when the iron was left in that state. I think most people will agree with him in saying a matte in gold assaying is bad practice, but I think there are cases when we can use the matte with certain precautions to great advantage, such as in the assay of battery chips, etc. The bisulphate assay is interesting, and I have no doubt yields excellent results, but as I have been very busy since getting the proofs of Mr. Jolly's contribution to the discussion, I have had no opportunity to test the method. I intend to do so, however, at my earliest opportunity, and until I do so hardly feel justified in criticising it at all.

Turning now to Mr. Whitby's contribution, there is not much to be said. It seems a pity that he had not tried a few experiments himself before he passed any criticism on the method. A little experience often makes us take quite another view of things. Taking the criticism from the beginning, we have just a brief reference to the functions of the ingredients of the flux I used. I think Mr. Whitby is entirely wrong in considering that the sulphur, iron, nail and charcoal all act as reducers. To begin with, the reactions are divided—to my mind—into two groups. First, we have a group consisting of the metallic iron of the charge, the sulphur and the iron nail, and the second group of the rest of the ingredients of the charge. The second group act much the same as in fluxing an ordinary banket sample, whilst in the first the sulphur mattes the metallic iron, the iron merely acting in the capacity of a safeguard against an excess of sulphur, which would give a lead sulphide. Should a lead sulphide form, it would pass into the matte and carry gold with it. The formation of this lead sulphide is what we have to guard against. Lead in the slag is reduced to metallic lead and passes into the button, so the litharge used is regulated according to the button required. I think Mr. Whitby's fear of retaining gold in the slag by an alkaline sulphide is hardly necessary, as the temperature of the fusion and the final wash of litharge and charcoal should overcome this little difficulty. An iron nail, melting into the lead button, is an experience I have not met with up to the present, and I should think the use of an iron rod of more ample proportions would overcome this difficulty. I think, in referring me to Peter's observations on the smelting, he could not have just compared in his mind the two processes of assaying and smelting in a blast furnace. As I remarked in my paper, the one essential point that has to be kept in view is to keep the lead out of the matte. Now in smelting lead ores carrying small proportions of copper, is this done? In the smelting of such an ore the copper bearing matte always contains very considerable proportions of lead sulphide, and so naturally carries values. I have no doubt the smelters would keep all the lead out of the matte if possible, but the fact remains that it is always there. It is unfortunate that Mr. Whitby raised the comparison of the two processes, as the conditions are such as to make the comparison useless. I would like, in conclusion, to thank those gentlemen who took part in the discussion.

EXPERIMENTS IN FIRE ASSAYING AT THE REDJANG LEBONG MINE, SUMATRA.

(Read at September Meeting, 1907.)

By G. B. Hogenraad (Associate).

REPLY TO DISCUSSION.

Mr. G. B. Hogenraad: In writing the paper it was not my intention at all to show how good the new flux was; as Mr. Guthrie remarks, the ore is simple enough, and in fact little trouble was experienced in applying the new flux. The intention really was to find out the effect of borax in assaying. I have always been taught that borax carried silver into the slag, and as far as the experience in assaying in our laboratory shows up till now, I cannot see any reason why borax should be added. The slag produced without borax contains no gold and sometimes but traces of silver, so that no cleaning of the slag is required; the lead buttons separate out very well, without slag adhering to them; the results agree satisfactorily, both for gold and silver.

Mr. Guthrie, and, in fact, everybody contributing to the discussion of the paper, agree with me that the old flux was not a good one for this ore; my point was that the fault lay in the excessive presence of borax, and I therefore regret that no one of my critics submitted a statement of results to show that borax had no detrimental effect, with especial reference to the silver. In a country like South Africa, where the silver values are, as a rule, of little importance, I can understand that the addition of borax does not make much difference. I should like to ask Mr. Guthrie to make experiments on 1 A.T. of ore with the flux he recommended on p. 75, viz., soda, 1·5 A.T.; PbO, 1·0 A.T.; borax 0·5 A.T.: and to compare the results with others, when using a flux consisting of: soda, 1·5 A.T.; PbO, 1·5 A.T. I am inclined to believe that he will find that there is a perceptible difference in favour of the latter flux, and if there were practically no difference, why not leave the borax out altogether, as it is rather an expensive article?

I made a good number of comparisons by using the two methods on the same samples and found that the figures, when using the scorification method, were always perceptibly lower than with the pot assay, both for gold and silver (see table); I therefore believe that the pot assay is the best, but that when scorifying and cupelling the buttons, checks must be used for the silver.

Mr. White says that the difference of 5% less PbO and 5% more soda does not account for the effects observed on the pots. I should like to believe him, but I cannot help thinking these pot crackings were chiefly due to excess of PbO, as after its decrease no more pots cracked. The pots have always been stored in the furnace room all day before use and stand five to six fusions. A cold pot has never been placed in a hot fire, as Mr. Guthrie supposed, and the makers are the Morgan Crucible Co., Battersea, England.

Prof. Stanley's suggestion that the lower yield of precious metal with the old flux was influenced by the excessive use of charcoal I readily accept; undoubtedly, there was too much charcoal, but does that prove the necessity of borax in the new flux? Another important question besides that of borax is:—Why should our buttons, occurring with such an oxidised slag, need scorification? To tell the truth, I cannot give an explanation for that; I can simply state the fact, that when they are not scorified before cupellation, the results are not so good. If this is indicative of imperfect fluxing, which I honestly believe it is, I should like to ask Mr. Whitby and Mr. Wilmoth what remedy they could suggest? Most probably they would advise me to add borax to the flux, but then we return to our starting point. I would state, for their guidance, that scorification of the lead buttons is not adopted to get agreement between duplicate assay of the gold but for the silver; and I would also mention that selenium has never been found in the lead buttons.

To Prof. Stanley I wish to express my regret at being not clear enough, when stating the amount of lead reduced by 1 gm. of charcoal. In fact, I never found more than 25 gm. for 1 gm. of charcoal, so that the sentence referring to this question should read:—"This was verified by several trials, always resulting in 34 to 35 gm. for 1 gm. of charcoal, when calculated for pure carbon."

Mr. Wilmoth is wrong in saying that I am inclined to blame the borax as being the cause of "excessive" loss during fusion. I never said so, and as regards the manganese, I hope he is convinced now by the following table, showing assay results obtained by pot assay and scorification, that there can hardly be any influence from that, and that scorification only is not the way to assay our samples.

Before ending this reply, I wish to express gratitude to those contributing to the discussion of my paper. I am quite willing to use borax in the flux, and will do so as soon as I have been convinced by figures, results from comparative experiments. Until then I will stick to my non-borax flux.

COMPARISON OF ASSAY RESULTS AS OBTAINED BY POT ASSAY AND SCORIFICATION.

Sample.	Date April, 1908.	Pot Assay Dwts. Gold.	Pot Assay Dwts. Silver.	Scorification Dwts. Gold.	Scorification Dwts. Silver.	Remarks.
Slimes	5	15·5	129·0	14·0	110·5	Pot assay carried out with non-borax flux; buttons scorified and cupelled. 2 A.T. sample taken.
Battery pulp	5	15·0	105·0	12·8	92·2	
Sands charge	5	11·4	81·6	11·1	78·4	
Sands charge	5	12·2	85·8	11·7	75·5	
Slimes	6	16·7	166·8	15·2	154·0	Scorification carried out in 2½ in. scorifiers with 50 gm. granulated lead and little borax to cover.
Battery pulp	6	20·6	115·9	17·5	142·0	
Sands charge	6	17·8	110·7	15·2	116·7	
Sands charge	7	21·8	163·2	18·7	152·8	
Slimes	7	20·8	189·7	16·3	162·6	
Battery pulp	7	21·5	164·5	17·5	144·4	
Average		17·33	137·52	15·00	122·91	5 gm. sample taken.
In favour of pot assay		2·33	14·61			

NOTES ON RAND MINING.

(Read at March Meeting, 1908.)

By TOM JOHNSON (Member).

DISCUSSION.

Mr. Tom Johnson (contributed): I have just received the March copy of the *Journal*, and I find that, by an oversight, there are a few mistakes in my paper, which I hope you will correct in the next issue:

Page 256, 2nd column, line 6 from bottom, read "1896" instead of "1906."

Page 258, 2nd column, line 4 from top, read "prevent" not "insure."

Page 259, 1st column, lines 6 and 7 from top, read, "per ton handled, as the amount is small. The value, etc."

Page 261, 1st column, line 8 from top, delete "for" before "a 200 ft., etc."

Mr T. Lane Carter: I regret that the hour is so late, as I had intended to deal at some length with Mr. Johnson's paper. With regard to what he says about shafts, I think that very few of the shafts here are worked to their utmost capacity. They can handle a great deal more rock than they do at present. Perhaps his criticism that there has been too much money expended on the size of shafts is well founded.

In regard to the author's remark about handling water in baling skips, it is interesting to note that on some of the large mines of the world baling has been adopted with success in preference to pumps.

The author refers to the common practice on the Rand of having a large number of shafts with boilers, etc. I think if we had adopted in the past, and if in the future we go in more for electricity on the mines, that this objection will not be so well founded. It seems to me that the small use of electricity on the mines is one of the things which we have to be blamed most for. We have in the past fought a bit shy of electricity, and I must say that if, in the future, we go in for more electricity, the results will be much better than they have been in the past. With regard to the use of telephones on the mines, this, of course, is not a very expensive item, and in some cases it is warranted, but those of you who have seen how expert the skipmen and engine-drivers become in the use of bells, will realise that a great deal can be done with bells. They seem to get a code of their own, and you can almost tell the temper of the man from the way he rings the bell. They seem to have the power of holding quite a conversation with these bells, so that the use of telephones in small shafts is by no means necessary. A good electric bell system is quite ample in the majority of cases.

The author asks, are we right in keeping to the rectangular form of shafts? I say, yes. This is a much better form for our conditions, it seems to me, than the round shaft. One proof is that there are only two or three round shafts on the Rand. We have had a considerable amount of shaft sinking on the Rand: we have had engineers from all parts of the world, and if round shafts had been better for the Rand we would have found it out long ago. I need not enter into a discussion of round versus rectangular shafts. The extra cost of sinking rectangular shafts and the utilisation of all the space will readily come to your minds, so that I think that the answer to Mr. Johnson's query is that rectangular shafts are preferable to round ones.

Bends in shafts are, of course, a considerable nuisance, but, as a rule, they are necessary. I think where the mistake has been made has been in having the radius of the bend far too small. The general practice has been to allow 50 ft. It seems to me that it ought to be at least 75 and possibly 100 ft.

His idea of endless rope haulage in the mine is a novel one, and under some conditions where we have flat stopes or flat reefs, as we are finding them on the East Rand, I think the scheme of endless rope haulage will play a most important part in the future. It is a common practice in coal mining, and the next ten or fifteen years, I think, will see this endless rope idea used a great deal in the opening up of deep level mines where the dip of the reef is small.

Mr. Johnson has given us some startling ideas, and one of them is the use of boilers underground at the bottom of the shaft. It seems to me that electricity has been proved to be so efficient and such a safe thing to use that the idea of having a steam plant underground is out of date. It might answer in collieries, but it seems to me it is out of the question on the Rand. By far the most startling scheme proposed in this paper is the placing two shafts close together. I suppose that he would take a property of 5,000 ft. long, and he would put his two shafts a few hundred feet apart, and you can figure it out what that would mean in developing the property. This is another coal idea. In fact, there is a great deal of the application of coal mining to Rand mining in this paper. The idea of those shafts close together is, of course, quite different to our present practice. The idea now is to get them as far apart as possible, and reduce the number of shafts sunk. There is, of course, difficulty in getting the connections through, but the final result is much better by putting the shafts as far apart as practicable.

The running of a 200-stamp mill from three levels is an excellent idea where it is possible, but there are not many mines on the Rand where this can be done. Only one or two mines in the Central Rand can run such a mill from three levels.

The author's ideas of stoping are sound on the whole. He seems to advocate having a gang of blasters, but I do not think that is the best policy. The contractor should have a direct interest in all the stores used, and that interest should not be spread over several parties. It should be to his interest to be economical in stores, where you have two people taking part in blasting, the greatest economy in stores will not be induced.

The idea of using tailings in the mine has already been discussed, and the fatal objection has been pointed out that the acid in the mines would produce hydro-cyanic acid. The idea is sound if it were not for this objection.

He refers to narrow stoping widths. Of course, it is the best practice to keep the stopes as narrow as possible, but with machines the problem is how to keep up the slabs of waste rock.

The author goes in for cheap hand steel. Here I entirely disagree with him. It seems to me that cheap steel is almost invariably the most expensive in the long run. I would never dream of buying some of the cheap steel we have on the market to-day. It is much better to get a good quality of steel, even if the first cost is high.

There is a point that will come up in the future, and that is the sharpening of steel underground. I think that in a few years hardly any of the steel that is now being hoisted to the surface for sharpening will be brought up. It will be done underground by electric furnaces or some other such scheme.

The author does not mention it, but there is difficulty experienced all over the Rand in regard to the loss of steel underground. Kafirs and Chinamen have a way of hiding steel, and on every mine you will find that there are tons of steel stored away underground and lost for ever. There are many ways of remedying this, but one of the best I think is to treat the hand men as we do the machine men—that is, to give them all a certain number of drills with a distinctive mark on them, and make the miners responsible. If any of the drills are lost, make them pay for them. This scheme has been tried on several mines, and proved to be successful. The expense at first is considerable in getting out sufficient steel, but eventually the saving is considerable.

Another point is machine sharpening versus hand sharpening of steel. It has been my experience that hand sharpening pays best. I have tried both methods.

With regard to guides, I think that in future, guides, to a great extent, will be of steel.

With regard to pumps, of course Cornish pumps have played a great part in mining in the past, but I think, to a great extent, the day of the Cornish pump is over, and that in the future pumps will be electrically driven.

A very important point is the use of gelignite instead of gelatine. The practice is in use on the Rand, and where gelignite can be used it is cheaper, and there is a very considerable saving in stoping. There is only one more point, and that is the extended use of white labour underground. The author seems keen on getting more white labour underground. The Mine Managers' Association has had a scheme on foot for a long time of apprenticing youths to underground work. The scheme has not been a success so far. At the present time there are very few apprentices at underground work. We have found that the principal trouble has been this. You know it is the custom on the Rand to take a young man and put him with a driller, a machine man for a few months, and at the end of that time, if he passes an oral examination he can get a blasting certificate. He leaves that mine and goes to the next, calling himself a full-fledged miner. The majority of these men in the past have obtained this certificate in six months. Now these youths are saying why should we be apprenticed for three years and go through the mill, when we can get a certificate and become a full-fledged miner and get £1 a shift! Of course, this reasoning is sound, and this is one of the reasons why this scheme has not been a success. One way to meet it would be to have two forms of blasting certificate. In the past there has only been one form. In the future there will be two forms. The man who comes to your mine knowing nothing about mining and who works for six months and gets a certificate, on his certificate will be these words, "this certificate gives no guarantee of the mining ability of the holder." Under these circumstances the manager would know the man was not a skilled miner. This new idea, which the Government has agreed to, may, to some extent, remedy the trouble. This is a most important point in regard to white labour underground, and we must all keep pegging away at it, because the more white youths we induce to take up mining on the Rand, the less will this great industry be at the mercy of miners from oversea.

WESTRALIAN WET-CRUSHING PLANTS, WITH SOME NOTES ON LABOUR EFFICIENCY.

(*Read at February Meeting, 1908.*)

By Gerard W. Williams, F.C.S., A.I.M.M.

DISCUSSION.

Mr. H. T. Brett (contributed): Having been absent from town for some time, on my return I was interested to note, while reading over some of the past months' *Journals* Mr. Gerard Williams' article, in which the author shows that he has given the subject much thought and attention. On the other hand, I was more astonished than interested to notice in the discussion on the above paper in March, some remarks made by Mr. H. A. White, which I think are rather overwhelming. One statement he makes is that the microscopic tube mill of 3 ft. 3 in. in diameter by 13 ft. long should be relegated to the laboratory (presumably, I suppose, to grind up chemicals, samples, etc.) and then, he continues, we should hear less of outputs of 27 tons per diem at a cost of 2s. 6d. per ton.

I would like to state for general information that the grading of sands on which this tonnage is accomplished is about as follows: 50 on 40, 35 on 100, and 15 on 150 mesh. The 13 ft. mill reduces 27 tons of this product to − 150 mesh per diem. As this mill takes approximately 20 h.p. and a 22 ft. mill between 50 and 60 h.p., we can presume that three 13 ft. tube mills are equal to one 22 ft. Therefore, the 22 ft. mill to be as economical as far as horse-power is concerned should take over 80 tons of sand per diem of the grade given above, and reduce it to a slime (− 150 mesh) at a cost of 2s. 6d. per ton. Personally, I have no idea in the difference in work accomplished by a 13 ft. and 22 ft. mill on Rand ore, but as Mr. White speaks with authority on the 13 ft. mill, I presume he has given the two machines ample attention to warrant the statements he has made. I am quite aware that the cost of regrinding on the Rand is about 6d. per ton milled, but as this only relates to grinding the coarsest of the sands to a finer state, it is obvious that if the whole of the sands (+ 150 mesh) were reduced to a slime (− 150 mesh) the cost of grinding per ton milled would rise considerably, and the cost based on *per ton of sand* ground to a slime would be somewhere in the neighbourhood of that given in the cost of a 13 ft. mill—2s. 6d per ton. It is well to differentiate carefully between the cost of grinding per ton milled and the cost per ton of sand ground to a slime. At the conclusion of Mr. White's remarks, he expresses astonishment at the high cost of treatment and low extraction obtained on the Kalgoorlie mine mentioned (Golden Horseshoe). While recognising fully the excellent work done on the Rand in obtaining 95 extraction at a cost of 5s. per ton, I would like to know what would be the figures if telluride (which exists in Kalgoorlie ores) suddenly made its appearance on the Rand, and concentrating, roasting concentrates, all-sliming, bromo-cyanide, and other luxuries had to be indulged in? If Mr. White could still, under these circumstances, obtain an extraction of 95 at a cost of 5s. per ton, or even more, then I should say he has Kalgoorlie—in fact, the world—at his feet.

Personally, I have found in most cases that it is fatal to generalise on mining between two different countries.

SOME FEATURES OF SILVER ORE TREATMENT IN MEXICO.

By W. A. Caldecott, B.A., F.C.S. (Member).

DISCUSSION.

Mr. S. H. Pearce (contributed): Like many others, I must also express my appreciation of the matter contained in this paper, not only for its own intrinsic worth, but for the reminiscences of the time when I paid a visit to Mexico myself, in 1904, and had a small experience in the cyaniding of silver ore. At that time, however, I was much more concerned in the extraction of the gold contents of the ore I was working upon, and which, being of far greater value than the silver contents, were at that time of more local importance. This, I might add was in the early stages of fine grinding, and my attention being concentrated on this, I was able to give less time than I would have liked to a matter that has since proved to be one of the greatest importance in that country. To illustrate my point, when one has a residue containing 4 dwt. of gold, a question of 2 oz. or 3 oz. of silver is of less importance commercially.

At the time of my visit, however, my curiosity was aroused by the fact that they were using a small quantity of mercuric chloride in their solutions, which gave them better extractions as regards the silver contents, and, I believe, had no injurious effect on the gold extraction. The idea, I understand, was first introduced by Mr. E. M. Hamilton, a well-known metallurgist working for one of our Past-Presidents, Mr. Chas. Butters. Well, as may be supposed, I worked out innumerable equations to account for this, and finally came to the conclusion that it acted in the dual capacity of a weak oxidising agent, and a "desulphuriser," if I may use the term, in that it was first reduced to a mercurous chloride, and as such combined with and rendered innocuous the monosulphides that were probably formed. I say "probably," because the ore was oxidised and apparently free from all sulphides. I could therefore only surmise that the silver contents existed as sulphide, and on that supposition I obtained some native argentite from a neighbouring mine, and carried out a few rough experiments, more, I must confess, because the subject worried me, than on account of any scientific interest, and I feel a keen regret that I have unfortunately lost the notes I made at the time, the more so as I intended them to form the basis of a paper for your benefit. The experiments were intended to note the effect of varying strengths of KCy solution, the addition of various reagents to these solutions, and also the effect of aeration as demonstrated by the mineral being floated on the top of the solution as compared with being submerged. The argentite was picked out from some specimen ore kindly supplied me, and crushed to an impalpable powder. The experiments were carried out in porcelain cups, and the effects judged by the time taken to effect solution. The results, which I have to quote from memory, were to the effect that, generally speaking, the rate of solubility, when submerged in fresh KCy solution, varied as the strength of solution employed, right up to the strongest solution used, viz., 5%. When floated, the action on the ore was much more rapid, and the effect of strength of solution not so marked. This pointed to the conclusion that oxygen was more important than strength of solution.

The next step was to try the effect of various reagents, which included all those available at the time. Of the many tried, the results of but few are worth recording. Oxidising agents produced an acceleration only on the submerged experiments, as might be expected, but the crux of the whole question was observed in the behaviour of the salts of the metals of the first group, viz., mercury and lead. Of the two, mercuric chloride gave a slight acceleration in point of time over lead acetate, particularly in the submerged trials, and this pointed to a reduction of the salt used. The experimentalist loves to theorise, and upon the slender basis of the foregoing, I formulated several theories, all quite satisfactory to my mind, and I shall leave it to others to shatter them as they will. In the first place, it appeared obvious to me that something was necessary to combine with the sulphur set free by the solution of the silver sulphide in an insoluble form. I attributed the superiority of the mercury salt to the fact that the mercurous sulphide formed, was the more insoluble compound, and therefore more stable, but the effect of the lead salt puzzled me for some time, until I hit upon the probable explanation that in conjunction with aeration it was converted into sulphate, also a stable and insoluble compound, and that the time required for oxidation represented the difference in activity between the two reagents. Upon these results I recommended the trial of lead salts as a substitute for mercuric chloride on a practical scale, as being less expensive, and though I was unable to stay to see the results, its general adoption since seems to warrant the assumption made at the time.

I am afraid I am rather digressing from the point, but it all leads up to my reasons for thinking that "reaction (*c*)" is incomplete. It was my most laudable intention on my return here to complete my investigations on the subject, but I found myself at once fully occupied with tube milling. It may be a debatable point whether lead sulphide is converted into sulphate, but I think it far more likely than that it should be converted into oxide as is assumed. Besides which, there is a necessary consumption of lead salt beyond that required by replacement of solution. The fact that practically no lead is precipitated on the zinc does not affect the question, as if present as sulphide it will not be in solution, and if as sulphate it would not be decomposed, whereas if present as oxide it would soon pass into solution and be precipitated on the zinc.

In conclusion, I would state that these theoretical considerations are always most fascinating, particularly for the reason that with the extremely dilute solutions of the reagents in question, accepted results, as per text book, are always likely to be upset. We have to thank Mr. Caldecott for his able demonstration of the case, and for pointing out its analogy to the work we are trying to improve upon on these fields.

Prof. G. H. Stanley (contributed): I read Mr. Caldecott's paper with great interest, because it was almost the first account I had seen of modern practice in Mexico, and I think the Society is indebted to him for bringing his observations before it, as they cannot be found in any text book. May we hope it is only the forerunner of a series dealing with other metals and processes which he saw during his journey.

The treatment described is apparently being used on ores similar in composition to those formerly treated by the Patio process, but of a lower grade, *i.e.*, containing native silver, haloid salts, and sulphide; but other ores occur carrying silver largely as sulpharsenide, sulphantimonide, and in other complex minerals, which would not yield a good extraction on the Patio, unless previously roasted with salt. Can Mr. Caldecott tell us whether these ores are being treated by the cyanide process as outlined, or by any other process?

Since the paper was read, two or three articles dealing in detail with the same subject have appeared in the technical press, one author particularly pointing out that silver metallurgy in Mexico is in a transitory stage, and from the accounts of the various plants given, it seems that agitation by compressed air in vertical tanks is now specially favoured and allows of the treatment thus of even 40 mesh material with low cost of power, these tanks are variously called Pachuca tanks, because first installed there, or "Brown" tanks, after their originator F. G. Brown, Waihi, New Zealand. New tanks are 13 ft. diameter and 55 ft. high. Time of treatment by agitation also is increasing and varies from 25 up to 150 hours, and one plant proposes, instead of selling its concentrates, to slime and treat by air agitation in strong cyanide solution, experiments having given satisfactory results.

It is very interesting to note the large employment of various types of suction filters, necessitated by the very small proportion of solution removed at each decantation and consequent imperfect washing and long time of treatment. For instance, on one plant the treatment tanks are 10 ft. deep. After agitation for twenty-four hours, the charge is allowed to settle six hours, and then 3 ft. only of solution is decanted off: then filled up with fresh stock solution plus lime and lead acetate and agitated till mixed for half an hour, then settled half an hour to 18 in. depth and decanted. Six other similar washes are given and two washes with water also, making nine decantations in all. Finally it is pumped to tall settlement tanks, where it remains as long as possible, and here the clear solution is decanted and the residue, running 70% moisture, discharged. Here it is proposed to instal filters, in order to save time and recover more dissolved silver and cyanide. At the works from which this instance was taken—Hacienda de San Francisco—crushing is done in ·025% KCN solution with 1,050 lb. stamps through slot screens, equivalent to 50 mesh, duty being 3·3 metric tons per twenty-four hours.

By Wilfley tables and Frue vanners, 2½% by weight of concentrates, are taken out of the pulp, which concentrates contain 50% of the values in the ore.

The tailings are separated by classifiers into sand 45·8% and slime 51·7% of the dry weight of ore, and treated by percolation and agitation respectively. Sands getting 14 days' treatment and slimes as above.

The weight of solution precipitated per ton of pulp treated and the strength of solution used are as follows:*

The sand treatment of 7 tons of ·35% KCy, 1·4 tons of ·10% KCy, and ·7 ton of wash water; in slime treatment 6·4 tons of ·10% KCy. Recovery 87%.

RESCUE APPLIANCES: LESSONS FROM GLENCOE.

(*Read at March Meeting, 1908*).

By H. Kestner.

DISCUSSION.

Mr. Tom Johnson (contributed): If, as our President says, this is the one that was used at the Courrières mine disaster by the men from Westphalia, I would like to point out that those men were a specially trained band. Now, on the

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* Bernard Macdonald, *Engineering and Mining Journal*, Apr. 4, 1908, p. 710.

Rand we want something that needs very little training to be able to use it in the gold mines: if a man was lying gassed in a winze or drive, there would be very little hope for him if he had to wait for one of these trained men to go and get the apparatus before he could be got out and taken to the fresh air. What we have to get, to enable a rescuer to get at a gassed person quickly, is something that even a Kafir can use and can also be kept underground close to where driving, winzing, etc., is being done. Being interested in these things, I have been on the lookout to see if I could find something that could be fairly handy and not need trained men, I have come to the conclusion that the smoke helmet and bellows is as good as most rescue appliances for this special purpose of getting out gassed men. I send you a cut of such an one,* which ordinarily requires two persons, one using the helmet and the other the bellows, and I fancy it would not take many minutes to make even Kafirs understand how to use it. A smart man could manage himself where compressed air was in use, by fitting the hose up to connect to the air pipes. This would not be so handy in coal mines giving off explosive gases, although the bellows supplied are good to feed the helmet through 200 ft. of hose. For greater distances a stronger pair of bellows would be necessary. For fire work, either in gold or coal mines, the liquid air apparatus, called the aërolith, would be very good; this is easy for men to handle and use, there being no taps, etc., for the man to look after. It is cool to wear and is not a great weight. I saw the aerolith at the Cockerill Collieries in Belgium, and they assured me there that they had adopted it after having tried several others, and this since the disaster in the North of France. They have a very complete rescue station there with their own air liquifying plant. Every mine would not need a liquifying plant. A small plant would serve ten or a dozen mines grouped together, and it would not cost an excessive amount per mine to instal a very complete equipment. There is a great advantage in the aerolith over most others, and that is, if men using it have to stay at a fire or in gases a longer time than the one filling of the apparatus will last, fresh supplies can be brought and emptied into the knapsack at the place without the men having to take it off, of course. The person bringing fresh supplies will be using the apparatus also. Another advantage is that the user gets cold air to breathe and is kept cooler than when using some others, and is therefore able to stick to the work for a longer period.

The meeting then closed.

Contributions and Correspondence.

THE INCIDENCE OF METHODS OF PAYMENT ON THE EFFICIENCY OF MINERS.

May I offer a somewhat belated contribution to the discussion on Mr. Kenneth Austin's paper. I regret that owing to the migratory habits which I have lately acquired the *Journals* only arrive at intervals.

In Broken Hill (New South Wales), all the mines pay their contractors by the actual weight of ore broken. Every truck of ore, as it leaves the ore pass, has a label attached to it, bearing on it the number of the party. Arriving at the brace, the trucks are run over a weigh bridge and the weight recorded by a clerk in the employ of the company, and checked by a tally clerk paid by the men. Every now and then a truck is tipped on to a sorting table, examined by the ore inspector and reweighed after any mullock has been thrown out. Miners who habitually send up mullock are dismissed. Dismissals from this cause are not very common. The method works splendidly, and no complaints are possible, as a representative of the men themselves checks every truck. All ore broken and nearly all tramming is done on contract. I know that this method would be difficult of application on the Rand, where self-dumping skips are in use, but it is the only method of paying contractors which is free from error. Every surveyor knows the frequent complaints, that are made as to the measurements on pay day. This liability to error is greatly increased in wide stopes. The Broken Hill stopes average 65 ft. in width, there are many stopes, however, where the width exceeds 200 ft.

Mr. A. Richardson mentions the failure of the Arbitration Act. That Act promised much and accomplished nothing. If anything, it accentuated labour troubles. Strikes are just as frequent in states possessing the Act as in the other states. In point of fact, the Act has caused more strikes and class bitterness than any factor of recent years. I do not know whether the consideration of the working of this Act comes within the province of this Society, but it seems not unlikely, that it may attract some attention in the near future, for I understand it is one of the projected measures of your new Government. If, however, the President considers that some information as to the way it has worked, or rather failed to work, in Australia, would be of interest to members, I should be more than pleased to submit some notes dealing with it. Out here labour politics form no small portion of the manager's worries. What with Arbitration Courts, Excise Courts, Wages Courts, Tariffs and other things, we are compelled to maintain a keen interest in current politics.

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* Omitted from letter.

Gerard W. Williams.

THE USE OF FILTER-PRESSES FOR CLARIFYING DECANTED SLIME SOLUTION.

The tonnage of ore crushed at the Knights Deep having increased, it was found necessary to instal more solution clarifying arrangements. A filter-press containing 36 hollow frames and served by a 3-in. high lift centrifugal pump was erected. The frames are 30 in. square inside measurement, and contain 450 sq. ft. of filtering area. The hollow frames admit of a cake of slimes 1 in. thick being formed before the press is full.

In the first instance the pump suction was connected directly to the decanting main. It was found with this arrangement that the cloths had to be changed at least every twenty-four hours and sometimes oftener, as they soon choked and the pressure rose beyond the capacity of the pump (40 lb. per sq. in.). The effect of steam for two hours to dehydrate any colloidal silica present was tried, and this increased the life of the cloths by anything up to twelve hours. Heated air should serve even better. The press was never by any means full of solids when it became necessary to change the cloths, and hence the necessity for so thick a hollow frame in this class of work is by no means obvious. Half the thickness in pressed steel or sheet iron should be sufficient, and inlets and outlets of solution into the press should be ample and of equal area. The cloths very soon became hard, and the pores choked even after vigorous scrubbing in water. On analysis it was found that 90% of the hardening solid deposited on and in the cloths was carbonate of lime, and Mr. McArthur Johnston experimented with various reagents to discover the best means of removing this. He found that by treating with hot hydrochloric acid of 5%—10% strength, and rubbing to release the bubbles of carbon dioxide, which prevented further action, the cloths were not only softened but were apparently uninjured. This is now being done regularly on the working scale. It was suspected that a good deal of the carbonate of lime giving this trouble was that which is seen floating on the surface of any undisturbed slime solution, and that the pump being connected direct to the decanting main, not only drew in this but also some slime, especially when the last few inches of solution were being decanted. The suction of the pump was then connected to the sides of two of the ordinary sand filter vats at a point opposite the inflow and about 18 in. above the sand level, and a box was put round the hole open at top and bottom to prevent any floating carbonate of lime being drawn in. This arrangement allowed the carbonate of lime to become wetted and to settle on the sand, and was so successful that cloths now run three and four days without changing, clarifying up to 800 tons per day. At the Simmer Deep the decanted slime solutions will enter the centre of a 25 ft. clarifying vat with peripheral overflow like a slime collector, from whence they will be pumped into the presses.

W. R. Dowling.

Obituary.

The death is recorded with much regret of Mr. Charles Gluyas (member), which took place on the 7th June, from pneumonia.

Mr. Gluyas, who was only 43 years of age, had been on the Rand for about 20 years, and for the greater part of that period he held the position of general manager of the Jubilee and Salisbury Mines. There was a very large attendance at the funeral at Braamfontein on Whit-Monday. Mr. Gluyas was elected a member in December, 1902.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Colorimetric Determination of Iron in Copper Alloys.—"The method is based upon the red colour reaction given by ferric salts with a solution of salicylic acid in presence of sodium acetate and acetic acid, the blue or green colour due to the copper being destroyed by the addition of potassium cyanide; 0·2 gm. of the alloy is dissolved in the smallest possible quantity of strong nitric acid, and if tin or antimony be present, the solution is diluted slightly and filtered. Lead, if present, is removed as sulphate. The solution is treated with 20 c.c. of a concentrated solution of sodium acetate and 10 c.c. of a 2% solution of salicylic acid in glacial acetate acid, and then a 3% solution of potassium cyanide (a strong solution will not answer the purpose) is added gradually until the green colour has disappeared and the precipitate of copper cyanide has re-dissolved. The solution is then made up to a definite volume and the red colour due to iron matched by adding a standard solution of ferric chloride to a mixture of 20 c.c. sodium acetate solution and 10 c.c. of the acetic acid solution of salicylic acid diluted to about the same volume as the solution of the alloy. Results are given showing the accuracy of the method."—A. W. Gregory, *Chem. Soc. Trans.*, 1908, *93*, 93—5.—*Journal of the Society of Chemical Industry*, Feb. 2, 1908, p. 163. (A. W.)

THE EFFECT OF HIGH LITHARGE IN CRUCIBLE ASSAY FOR SILVER.—"The object of this paper is to point out that the use of a large excess of litharge in the assay of some ores will give results for silver that are uneven as well as low. So far as I know, however, an excess of litharge does not affect the results obtained in the crucible assay of ores for gold.

The main reasons for using, in the crucible assay, much more litharge than is required to give the necessary lead button are: (1) Its action as a flux; (2) its action as a desulphuriser; and (3) its action as an oxidiser, especially on metals like copper and nickel, whereby they are forced into the slag as oxides and thus prevented from passing into the lead button. Hence, by the use of much litharge in the crucible assay more ore can often be taken than in the scorification process, and a lead button obtained which can possibly be cupelled at once or after a single scorification. The method is specially advantageous with an ore carrying much copper or similar impurity and poor in silver, when the assayer does not wish to resort to a wet analysis for the determination of the silver. For several years I have noticed that, when much litharge was used with certain sulphide and arsenical ores, the results were considerably lower than when the scorification method was used. This fact was more forcibly brought out in connection with certain work carried on this year by H. A. Frame and F. C. Jaccard, students at the Massachusetts Institute of Technology, to ascertain the best method of assaying the rich arsenical nickel and cobalt ores from Ontario.

The following are some results obtained: Ore No. 2,687-2 had a reducing power of 4·2 and contained nickel, 12·92; cobalt, 10·92; and arsenic, 46% The minerals noticed in the ore were niccolite, smaltite, erythrite, cobaltite and arsenopyrite.

Charge for the Crucible.						Results.		
Assay, ton.	Sodium bicarb.	Borax glass.	Litharge.	Argols (R.P. = 11)	Lead button.	Silver found.		Silver in slag.
	gm.	gm.	gm.	gm.	gm.	gm.	Oz. p. t.	
1-20	10	5	30	1½	23	0·11615	2323	Not
1-20	10	10	35	1½	23	0·11648	2329·6	assayed
1-20	10	5	80	1½	28	0·11272	2254·4	52·4 oz.

Ore No. 2 687-6 consisted chiefly of smaltite, erythrite, niccolite and arsenopyrite: R.P. = 4·06; nickel, 3·94: cobalt, 11·25: and arsenic, 59·7%.

Assay, ton.	Sodium bicarb.	Borax glass.	Litharge.	Argols.	Lead button.	Silver in slag.
	gm.	gm.	gm.	gm.	gm.	Oz. p. ton.
1-20	10	10	35	1·5	22	258·2
1-20	10	10	80	1·5	21	241·2

The same ore assayed after amalgation:—

1-20	30	10	30	1·5	23	229·8
1-20	10	10	35	1·5	23	230·8
1-20	10	10	80	1·5	23	220·2

Subsequent to the above tests I made further investigations on the ores tested as well as on other ores, and obtained the following results: Ore No. 2,687-2; R.P. = 4·2; nickel, 12·92; cobalt, 10·92: and arsenic, 46%.

Scorification Method—Charge.					Results.
Assay, ton.	Borax glass.	Lead.	Silica.	Lead button.	Silver.
	gm.	gm.	gm.	gm.	Oz. p. ton.
1-20	3	65	1·5	11	2330·4
1-20	5	65	1·5	10	2338 0

Crucible Method—Charge.						Results.
Assay, ton.	Sodium bicarb.	Borax glass.	Litharge.	Argols R.P = 11	Lead button	Silver.
	gm.	gm.	gm.	gm.	gm.	Oz. p. ton.
1-20	10	6	35	1·5	22	2333·2
1-20	10	6	35	2	30	2330·8
1-20	10	6	80	1·5	23	2229·2
1-20	10	6	100	1·5	25	2244·2

Ore A. Chiefly smaltite and niccolite with free silver, containing nickel, 5·06. and cobalt, 9·12%.

Assay, ton.	Sodium bicarb.	Borax glass.	Litharge.	Argols.	Lead.	Silver in lead.	Silver in slag.	Silver in cupel.
	gm.	gm.	gm.	gm.	gm.	Oz. p. ton.	Oz. p. t.	Oz. p. t.
1-20	10	10	30	1·5	19	2051·4	9·6	34·0
1-20	10	10	40	1·5	21	2056·0	—	—
1-20	10	10	40	1·5	21	2050·0	—	—
1-20	10	10	80	1·5	30	1968·6	—	—
1-20	10	10	80	1·5	22	1944·6	135·2	35·0
1-20	10	10	80	1·5	21	1984·8	70·2	34·6
1-20	10	10	80	1·5	21	1914·8	—	—

Ore No. 2,703-1 consisted of smaltite, native bismuth and native silver in calcite; R.P. = 5; nickel, 0·3; cobalt, 8: and arsenic, 55%. The scorification assay gave 404·8 oz., and the combination wet and dry analysis gave 403·7 oz. of silver per ton.

Crucible Method—Charge.					Results.	
Assay, ton.	Sodium bicarb.	Borax glass.	Litharge.	Argols.	Silver in Lead.	Silver in Slag.
	gm.	gm.	gm.	gm.	Oz. p. ton.	Oz. p. ton.
1-20	10	6	35	1·5	402·6	Not
1-20	10	6	80	1·5	388·0	assayed
1-20	10	6	100	1·5	392·0	11·8

The same ore after amalgamation gave, by the scorification assay, 292 and 291·2 oz., while by the combination wet and dry analysis the result was 292·2 oz. of silver per ton.

Crucible Assays—Charge.					Results.	
Assay, ton.	Sodium bicarb.	Borax glass.	Litharge.	Argols.	Lead button.	Silver.
	gm.	gm.	gm.	gm.	gm.	Oz. p. ton.
1-20	10	6	35	1·5	24	293·0
1-20	20	6	35	1·5	23	292·8
1-20	20	6	35	1·5	25	291·8
1-20	10	6	60	1·5	24	288·0
1-20	10	6	80	1·5	26	283·6
1-20	10	6	100	1·5	26	279·8

I have a number of other examples, but these should be sufficient to illustrate the fact to which I wish to call attention. The ores used carried practically no gold. The same lot of litharge was used in all the fusions, and the conditions under which the fusions were conducted were as nearly identical as possible. Heavy crystals of litharge were found on all the cupels. Only $\frac{1}{20}$ assay ton of ore was used in the assays, because Messrs. Frame and Jaccard found that in case of ore No. 2,687-2, carrying 12·92 % of nickel, if 5 gm. of ore were taken and 35 gm. of litharge were used, the resulting lead button would not cupel. If 5 gm. of ore were taken and the litharge increased to 80 gm., in order to slag the nickel, the button would cupel but the silver results were low. Some ores from the cobalt district carry so large a percentage of nickel that $\frac{1}{20}$ assay ton of ore is the limit that can be used in either scorification or crucible work, and low litharge is inapplicable in the latter method. The following ore serves as an illustration: Ore No. 2,703-2, niccolite (NiAs); R.P. = 5·3; nickel, 38·01; cobalt, 1·19; and arsenic, 53·31%.

Assay ton.	Sodium bicarb.	Borax glass.	Litharge.	Silica.	Argols.	Ratio of litharge to n'k'l in ore	Lead
	gm.	gm.	gm.	gm.	gm.		gm.
1-20	15	10	120	5	1	214 to 1	·24
1-10	15	10	120	5	none	107 to 1	·25
2-10	15	10	120	5	nitre 1	54 to 1	·25

The lead buttons from $\frac{1}{10}$ and $\frac{2}{10}$ assay ton of ore would not cupel, both cupels being covered with a thick film of green nickel oxide (NiO). The lead button from $\frac{1}{20}$ assay ton of ore would just cupel, leaving the cupel stained green."—R. W. LODGE.—*Mining World* (abstract of paper read before Am. Inst. of M. E.), Feb. 1, 1908, p. 213. (H. A. W.)

Remark.—It is a pity the author still uses the long ago condemned bicarbonate of soda in place of the modern method of preferring the carbonate. [ED. COM.]

ANHYDROUS BORATES. — "The preparation of borates by fusing boric anhydride with the oxides, nitrates, and carbonates of different metals in a specially constructed furnace for three hours, at a temperature from 1,350—1,450°, is described. Many oxides like those of iron, aluminium, chromium, silicon, zirconium, tin, molybdenum, tungsten and uranium were found to be insoluble, or but slightly soluble in boric acid. Some oxides were reduced to a lower stage of oxidation, others were oxidised still higher. In tin compounds the stannic oxide was partially reduced to tin. Sulphates, chlorides, bromides, and iodides were found to be unsuitable in the preparation of the borates; and from alkali carbonates the carbon dioxide was difficult to expel. The oxides soluble in boric acid form three groups: (1) Those which, in equivalent admixture (valency of metal atoms = valency of boron atoms) produce clear, homogeneous fusions, either crystallising on cooling or leaving clear glasses without segregation. (2) Those which produce clear, homogeneous fusions at the higher temperatures but show dissociation and the formation of an emulsion at a definite degree of concentration. This is a case of limited mutual solubility of two liquids which are miscible at higher temperatures in all proportions. (3) Those which do not give homogeneous fusions even at 1,400°, two distinct layers of liquids being produced, like oil and water. The equivalent mixtures crystallise as orthoborates. To the first group belong the oxides of lithium, potassium, sodium, rubidium, caesium, thallium, and silver. Sodium and potassium metaborate, $NaBO_2$ and KBO_2, fuse at 820 and 890° and deliquesce in the air. They are remarkably volatile and dissolve alumina, chromic oxide and silica freely. To the second group belong cuprous oxide and the oxides of lead, bismuth, antimony, arsenic, titanium, molybdenum, and tungsten. They produce emulsions, but show no crystallisation. Some of the glasses are of different hardness: PbO,B_2O_3 is as hard as flint glass, while $3PbO,B_2O_3$ softened in boiling oil. The third group embraces the oxides of calcium, strontium, barium, magnesium, zinc, cadmium, manganese, iron, cobalt, and nickel. Magnesia produces two compounds, orthoborate, $3MgO,B_2O_3$, and pyroborate, $2MgO,B_2O_3$. No other anhydrous magnesium borate exists. The corresponding borates of cobalt and nickel were likewise found. It was not possible to produce the ferrous borate, since oxidation invariably took place. The fusion produced two layers, of which the lower showed crystals having approximately the composition, $3B_2O_3,2FeO,2Fe_2O_3$. In melting cupric oxide the length of fusion played an important rôle and two layers were produced. The lower layer showed crystals of CuO,B_2O_3, which decomposed on longer heating, evolving oxygen. On complete decomposition the residue again consisted of two layers, the lower of which contained crystals of $3Cu_2O,2B_2O_3$, which, on still further heating, decomposed into cuprous oxide and boric anhydride, forming an emulsion. Crystals of $3ZnO,2B_2O_3$ and $3ZnO,B_2O_3$ were obtained. The manganese boric acid fusion resulted in two layers which devitrified on heating and formed crystals of the formula, MnO,B_2O_3. Other borates, produced by previous investigators, are $3MnO,B_2O_3$ and $3MnO,2B_2O_3$. According to the author, the following compounds of the alkaline-earths with boric acid exist, of which the pyroborates have been first prepared by him.

Orthoborates	$3CaO,B_2O_3$	$3SrO,B_2O_3$	$3BaO,B_2O_3$
Pyroborates	$2CaO,B_2O_3$	$2SrO,B_2O_3$	$2BaO,B_2O_3$
Metaborates	CaO,B_2O_3	SrO,B_2O_3	BaO,B_2O_3
Biborates	$CaO,2B_2O_3$	$SrO,2B_2O_3$	$BaO,2B_2O_3$

The author recognises five types of anhydrous borates in place of the four of Le Chatelier: (1) $3Me''O,B_2O_3$, orthoborate: (2) $2Me''O.B_2O_3$, pyrobarate; (3) $3Me''O,2B_2O_3$, sesquiborate; (4) MeO,B_2O_3, metaborate: (5) $MeO,2B_2O_3$, biborate. With reference to devitrification, the author states that it is a process closely akin to an explosive chemical reaction, set up at a definite temperature, since an amorphous substance is as unstable a system as a mixture of hydrogen and oxygen."—W. GUERTLER, *Sprechsaal*, 45, 612—6.—*Journal of the Society of Chemical Industry*, Feb. 25, 1908, pp. 158—9. (A. W.)

LEAD SILICATES. — "Mixtures of rock crystal (99·96—99·97 % of silica) and lead oxide, in different proportions, were fused, and the products examined. Combination took place at 700°—800° C., in most cases below 750° C.: the products were solutions of lead in oxide in lead glasses or silicates. The solubility of the lead oxide increases with the temperature. On cooling the products, the dissolved lead oxide gradually separates, with evolution of heat. The characters of the products are shown in the following table:—

	Composition.	SiO_2.	Physical characters.	Intervals of:	
				Temperature of formation.	Softening and melting temperatures.
		Percent.			
1	$6PbO : SiO_2$	4·32	Dark brown, hard and brittle: dark yellow when powdered. Contains little 'glass.'	709–794 C.	740—789° C.; melts to a thin liquid; solidifies suddenly
2	$5PbO : SiO_2$	5·14	Lighter colour than No. 1: wax-like, and contains more 'glass' than No. 1. Resembles PbO when powdered.	727–798 C.	736—796° C.; melts, etc., like No. 1.
3	$4PbO : SiO_2$ (subsilicate)	6·34	Hard, brittle, umber-coloured mass. When powdered, is of lighter colour than PbO.	709–726 C.	719—729° C.; melts, etc., like No. 1.
4	$3PbO : SiO_2$	8·51	Hard, wax-like mass.	700–725° C.	714—725° C.
5	$2{\cdot}5PbO : SiO_2$	9·78	Light yellow brittle mass; greyish-white powder; contains much 'glass.'	708–732 C.	728—740 C.
6	$2PbO : SiO_2$ (singulosilicate	11·94	Dark amber-coloured glass; powder of lighter colour than No. 5.	718–720 C.	Begins to soften at 730 C.; thin at 765° C.
7	$4PbO : 3SiO_2$ (sesquisilicate)	16·89	Light-yellow, strongly refractive and iridescent glass; white powder.	709–720° C.	Softens at 740° C., and remains viscous.
8	$PbO : SiO_2$	21·32	Colourless crystal glass; snow-white powder.	709–724 C.	Softens at 740—750° C.

All the products, when finely powdered, are soluble in hot dilute nitric, hydrochloric and acetic acids, with separation of amorphous silica. The silicate, PbO,SiO_2, is also soluble in hot dilute alkalis with formation of alkali silicate and plumbate; from the solutions, nitric acid precipitates amorphous silica, whilst hydrochloric acid precipitates crystalline lead chloride and silica. At 350°—600° C., the lead oxide in the silicates is reduced by hydrogen first to lead suboxide, Pb_2O, and then to metal. (The reduction of free lead oxide by hydrogen begins at 290°—300° C. with formation of the gray suboxide; at 410° C. the reduced metallic lead collects in the form of globules.) The silicates can also be reduced by means of carbon monoxide or hydrocarbons.

These results indicate that in the roasting of galena, the proportion of silica in the charge is an important factor. When sulphide ores of lead are roasted in presence of silica at 700°—750° C., the lead oxide formed will combine with the silica. The silicate thus produced will dissolve further quantities of lead oxide, the solubility of the latter increasing with rising temperature. The silica will thus, at high temperatures, prevent the lead oxide from being converted into sulphate. This occurs, for example, in the slagging-roasting of galena in continuous reverberatory furnaces and in the Huntington-Heberlein and Savelsberg processes. In the cooler portions of the furnaces: for example, on the upper hearth of a furnace with two hearths, the temperature is not high enough for the formation of silicates, and the lead oxide is converted into sulphate by the sulphur trioxide in the furnace gases; the sulphur content of the ore-charge remains unaltered or is only reduced by 1—2%. This confirms the view that in the roasting proper, where the temperature rises to 700° C. and above, the silica exerts a very favourable effect. The separation of metallic lead in the furnace-charge frequently observed, is due, in part, to the reduction of lead silicates by reducing gases from the fire."—W. Mostowitsch, *Metallurgie*, 1907, *4*, 647—655.—*Journal of the Society of Chemical Industry*, Feb. 29, 1908, pp. 163—4. (A. W.)

Explosive Combustion.—"At the Royal Institution, on Friday, February 28, Prof. W. A. Bone gave a lecture on 'Explosive Combustion, with special reference to that of Hydrocarbons.' He said that the question of how a hydrocarbon burned, that was, precisely how it was attacked by the oxygen, had been the subject of much discussion during the past 15 years. The two combustible elements, carbon and hydrogen, of which a hydrocarbon was composed, were, in a sufficient supply of oxygen, ultimately burnt to carbon dioxide and steam respectively, but on kinetic grounds it seemed inconceivable that the passage from the initial system of, say, ethane and oxygen to the final system of carbon dioxide and steam could be immediate and direct. It had therefore been universally recognised that the process involved a number of successive stages, though opinion had been sharply divided as to their nature. During the greater part of last century the belief prevailed that the hydrogen was much the more combustible of the two elements of a hydrocarbon, and consequently that when combustion occurred in a limited supply of oxygen the hydrogen was preferentially burnt. This view was supported by Graham and Faraday, and for 50 years was regarded as one of the most certain articles of chemical faith, until it was finally overthrown by Dixon and Smithells in 1892. A second theory originated in 1861 with Kersten, who, as the outcome of experiments on the explosion of a mixture of ethylene and electrolytic gas, boldly asserted 'that before any portion of the hydrogen is burnt all the carbon is burnt to carbonic oxide, and that the excess of oxygen then divides itself between the carbonic oxide and the hydrogen.' Kersten thus attempted to substitute the idea of the preferential burning of hydrogen, but his views received no serious attention till they were revived and endorsed by Dixon and Smithells in 1892. But the idea of a 'preferential' combustion, whether of carbon or of hydrogen, seemed repugnant to well-established principles concerning the nature and conditions of chemical interactions in gaseous systems, and a third view was that the solution of the problem lay in the assumption of an initial association of the hydrocarbon and oxygen, forming an unstable 'oxygenated' molecule which subsequently rapidly decomposed. Many years ago Prof. H. E. Armstrong suggested that the combustion of a hydrocarbon took place under the conjoint influence of water and oxygen, and involved the successive formation of intermediate 'hydroxylated' molecules, which at high temperatures rapidly decomposed into simpler products; but little notice was taken of his suggestion at the time. Prof. Bone, however, said that in conjunction with his pupils at Manchester University he had carried out researches covering the entire range of conditions under which hydrocarbons could be burned, and an exhaustive study of the slow combustions of methane, ethane, ethylene and acetylene at temperatures between 250° and 400° C. was decisive against the preferential burning, whether of carbon or of hydrogen, the balance of evidence being decidedly in favour of the hydroxylation theory, with the proviso, however, that the oxygen was directly active. He then performed a number of the experiments, which he thought disposed of the theory of the preferential combustion of carbon, and remarked that they afforded no evidence of any real discontinuity between the chemical phenomena of ordinary inflammation and those of detonation. The higher temperatures and more violent conditions of detonation were responsible for the more complete breaking down of unsaturated hydrocarbons and a greater 'unburning' of steam by carbon, but there was probably no difference as regards the mode in which the hydrocarbon was attacked by the oxygen in the two cases."—*Times Engineering Supplement*, March 4, 1908. (J. A. W.)

METALLURGY.

Sand Collecting and Washing.—"Auriferous sand, when kept in close contact with continually renewed amounts of cyanide leach by stirring, is known to give off its gold far more readily than if it is allowed to remain at rest in the leaching vat, the liquid penetrating through it in a downward direction. Still, so far it has not been possible to design a stirring apparatus by the aid of which considerable amounts of ore could be kept in permanent motion and intimate contact with the cyanide particles with the expenditure of relatively low amounts of energy. Ores containing clay or other earthy impurities are especially refractory to permanent stirring, owing to their becoming partly glued together. In fact, such impurities are liable to prevent a normal leaching process of ground ore, the latter, as well as the sand filter, being made impervious, and the conduits and suction pumps becoming obstructed.

Difficulties of this kind were experienced by a Chilian mining company, whose relatively rich ores (20-30 gm. of gold per ton) were impaired by iron clay impurities aggregating as much as 30%. After extensive trials had failed to ensure a separation of the clay, Mr. Reinhold Freygang, of Hamburg, succeeded in designing an interesting apparatus for effectually removing those impurities and allowing the ore to be readily leached.

This apparatus comprises a slanting tube, inside of which a worm revolves, water and ore powder being supplied through a vertical tube and hopper. After sinking downwards in the water contained in the vertical tube the ore powder is seized by the worm, and while being stirred is thrown upwards, the clay is separated, and the water with these admixtures in suspension leaves the tube through the outlet. All heavier particles—that is to say, the washed-out sand—having been carried upwards by the worm, are dropped in a moist condition from the upper end of the slanting tube into the leaching tank, in which the normal vat leaching process can then take place. It may be said that the total amount of clay contained in the ore could be removed by a single washing, and as the water used in this connection is drawn back from the settling tank, in order to be used over again, it is by no means lost. The clay slimes precipitated in the settling tanks, and which still contain considerable amounts of gold, can then be treated with cyanide leach in a special battery of stirring worms arranged in series.

By arranging several units in series (the sand cleaned in the first apparatus being discharged into the hopper of the next) a real leaching battery is obtained provided the second hopper be fed with cyanide leach instead of with water. The leach issuing from the second apparatus then is either allowed to enter the hopper of a third apparatus or is carried away in order to be freed from its gold, this third apparatus being supplied with further amounts of leach. The same process can be continued until all the gold contained in the sand has been removed. The gold carried along by the clay discharged from the first apparatus then is separated in a similar manner.

In order to work the sand and clay simultaneously in two parallel batteries, the first apparatus, containing a mixture of sand and clay, may be fed immediately with cyanide leach instead of with water. This leach, along with washed-out clay impurities, then enters from the outlet into the hopper of the parallel clay leaching battery, whereas further amounts of leach are supplied in the next hopper to the sand issuing from the first apparatus at the end of the slanting tube.

This process obviously allows of a great number of variations to suit the requirements of each kind of ore. The power consumption is extremely low (an apparatus of 20 tons daily output being operated by a 1½ h.p. motor), while the stirring effects are most rapid and perfect.

The apparatus further lends itself to a number of applications in the chemical industry, being used to advantage whenever a fundamental substance is to be freed from clay or other impurities, such as in washing chalk or china clay (kaoline), preparing earthy dyes, or separating earthy compounds (chromium oxide, etc.) from the mineral.

According to whether the heavier or lighter portions of the material are to be subjected to further separation or treatment, an arrangement in parallel or in series of the various apparatus constituting the battery should be given the preference." —*The Times Engineering Supplement*, Feb. 5, 1908. (W. A. C.)

Tantalum.—"Owing to its rarity, tantalum and its qualities are comparatively little known: but, as in the case of the other metals treated in this paper, constant investigations in its metallurgy and properties are being carried on.

The one principal use to which the metal is now put is as a filament in incandescent lamps. These lamps were put on the market in Germany in 1905, and on the American market in 1906. There is a considerable saving in current in their use, as they consume but 2 watts per candle-power, as compared with 3·5 watts per candle-power for the ordinary carbon filament lamp.

An excellent article by W. von Bolton and O. Feuerlein, who developed the tantalum lamp for the Siemens & Halske Aktien-Gesellshaft, was published early in 1905, which gives the results of their investigations of the metal.

A portion of this article relating to the tantalum incandescent lamp was quoted by Dr. J. H. Pratt in the 'Mineral Resources of the United States for 1904.'

Some of the properties of this remarkable metal are given in the following condensed translation of a portion of the article cited:—

In the cold the metal is extraordinarily inert. The metal is not attacked by hot hydrochloric, nitric, or sulphuric acids, aqua regia, or alkaline solutions, but is attacked by hydrofluoric acid.

Heated in the air at 400° C. it becomes yellow, like steel, and like this, when heated to 600° C., or for a longer time at 500° C., it becomes blue. In fine threads heated in the air it burns with little intensity and small flame. It eagerly combines with hydrogen and nitrogen at the beginning of the red glow, forming metallic-looking but brittle compounds. It combines easily with carbon, forming several carbides, which, so far as known, appear metallic and are very hard, but brittle. From the high atomic weight of tantalum (181) it can be seen that a very small percentage of carbon (atomic weight 12) is sufficient to carbonise a relatively large amount of tantalum.

Melted and drawn tantalum has a specific gravity of 16·8. In powder, with oxygen and hydrogen still present, it has a specific gravity of 14. It is a little darker than platinum and of about the hardness of soft steel, with a greater tensile strength. It can be hammered into plates, though individual blows show little effect. It can be rolled and drawn into fine wire, having a tensile strength that is remarkably high, amounting to 93 kgm. per sq. mm., while that of good steel is 70 to 80 kgm. If a lump of tantalum heated to a red glow is placed under a steam hammer, it may at once be beaten into a plate, which, after being heated and hammered several times, attains a hardness equal to a diamond.

An attempt to perforate such a sheet 1 mm. thick with a diamond drill running continuously at 5,000 revolutions per minute for three days penetrated but ¼ mm., while the drill was much worn. A complete perforation could not be made. Despite this, the plate could still be rolled thinner—a remarkable union of toughness and hardness. If it were only a cheap metal, it would be ideal for armour plates. Siemens & Halske A.-G. hope to employ it for dies, drills, bearings, journals, etc.

At 110 volts, for a 32-candle-power lamp, a filament 700 mm. (about 27·6 in.) long, and for a 25-candle-power lamp a filament 650 mm. (about 25·7 in.) and 0·05 mm. in diameter is used. As the filament softens when highly heated, it cannot be used in a loop like a carbon filament, so that the arrangement was finally hit upon of looping it back and forth upon metal supporters held by a central glass rod.

For 220 volts a filament 1,350 mm. (about 48·2 in.) long is necessary and becomes impracticable. A filament 650 mm. long by 0·05 mm. thick weighs 0·022 gm., so that about 45,000 filaments will weigh a kilogram (about 20,600 filaments per lb.). A 0·05 mm. wire will support 400 gm.

This lamp gives a comparatively white light, and after it has been used for about five hours, it gives 15 to 20% more light, owing to the filament changing to the form of a series of semi-globular masses. There is afterwards a diminution of light, but the lamp's life is somewhat longer than that of the carbon lamp. When it burns out, its ends will weld if brought in contact under current, and the lamp will go on burning. It will stand a much greater overload than a carbon lamp, and with 50% overload shows much less blackening of the bulb for the same time. With rising temperature the resistance increases, while with carbon it decreases. It gives a steadier light than a carbon lamp, because it is less sensitive to current changes.

These lamps are now on the market in very small forms to be used on low voltages (2 to 5). The light of all the forms is excellent, approaching the tungsten lamp in whiteness.

In carrying out the manufacture of pens from tantalum metal, the metal having been smelted so as to obtain in the first instance a lump of metal of irregular form, which is in a soft condition, it is worked by rolling, or otherwise, in any suitable known manner into the form of thin sheets, from which the pens are produced by the same means as those employed for the manufacture of pens of other metals. By the said process of rolling the tantalum metal assumes, as before stated, a much greater degree of hardness than before, and it may therefore require to be repeatedly annealed between the several stages of rolling.

After the final stage of rolling, the thin sheets of tantalum metal may be found to be of the right hardness for forming the pens therefrom; if too hard they may be again annealed to a certain extent, or if the pens are required to have a greater degree of hardness than can be attained by merely working the metal, the tantalum may have a small quantity of the before-mentioned hardening substances, such as carbon, added to it. This may either be effected, in the first instance, when the metal is smelted, a very small quantity, such as 0·1%, of carbon being sufficient for the purpose, or the prepared sheets of tantalum metal may be packed in carbon powder and subjected to a considerable heat for some time.

If an alloy of tantalum with another metal is to be employed, the most suitable metals for this purpose are wolfram and iron, either of these metals being simply added to the tantalum metal when in the fused condition. Care must in this case be taken to protect the tantalum against chemical change, for which purpose the melting is preferably effected in vacuo. The composition of such alloys may vary very considerably, but very suitable alloys for the purpose consist of from 95 to 98% of tantalum and from 2 to 5% of wolfram or iron.

Tantalite, in at least a majority of the cases, occurs in pegmatites or coarse granites."—FRANK L. HESS, Mineral Resources of the United States, 1906. —*London Mining Journal*, Jan. 25, 1908, p. 117. (A. R.)

THE HOT BLAST IN PYRITE SMELTING.—"Mr. L. Parry is quite right in his contention that it is not at all necessary to have heated blast for *pure* pyritic smelting, while it may be quite advantageous for *partial* pyritic smelting or plain matte smelting where the costs of heating the blast are not prohibitive, and there are good thermo-chemical reasons for it.

In pyritic smelting proper we should burn iron sulphides (with small amounts of zinc sulphide, lead sulphide, manganese sulphide, iron arsenide, etc.) for fuel, and there must always be an excess of iron sulphide to go to the matte produced, as otherwise two grave difficulties will be encountered, *i.e.*:—

1. Unless the iron in the matte produced is over 30% of its copper contents, the slags produced with it will be of too high a tenour in copper to be thrown away, and will require re-smelting.

2. If the matte produced should have a copper contents exceeding about 75% of their weight, the explosive reaction between Cu_2O and Cu_2S, so well known in both reverberatory and converter practice, would take place and stop the operation.

Neither of these difficulties is likely to occur in ordinary practice, but I have known the former to take place for short periods in my own practice, and am told that the latter actually occurred in an experiment in Bisbee, Ariz., and they afterwards found matte of 76% to 78% copper high up in the charge when cleaning out the furnace.

If the matte produced contains iron in excess of, say, 50 % of its copper contents, it is plainly evident that there is an excess of fuel on the charge, as converter practice shows that such a matte will give off enough heat to smelt the lining necessary to form its slag and increase its temperature and that of the slag produced some 300° C., at the same time giving off in the gases produced about double the amount of heat contained in the matte when the blow commenced.

In the ordinary pyritic practice mattes with 12 % to 20 % copper and 50 % to 60 % iron are made from ores containing 2 % to 3 % copper. These mattes, therefore, contain a very large amount of fuel unburned, and carry away from the furnace the heat due to the temperature of the excess iron and sulphur which they contain, as well as the heat that should have been produced by its combustion. This is plainly shown from the fact that we are able to take these mattes after cooling and smelt them again in a similar manner with proper additions of fluxes and blast to a matte of a converting grade practically without other fuel than their own iron and sulphur contents. Metallurgists generally have recognised the fact that no carbonaceous fuel is necessary, and Mr. J. Parke Channing says that the coke seemed only to be necessary to keep the charge in a good mechanical condition.

Given the ores with the proper composition and necessary fuel value, the difficulty with pyritic smelting lies wholly in the nature of the fuel, together with the difficulty of securing the metallurgical ability and fine manual skill to burn the fuel in our present appliances. Our furnaces are constructed with the sole view to smelting with a solid fuel which cannot be melted (*i.e.*, carbon) while the fuel we should use in pyritic smelting melts at from 1,000° to 1,050° C. With a molten fuel (FeS) falling through the focus or smelting cone of the furnace, it is more or less a matter of chance whether you burn it or not; and in our present general practice we burn very much less than we would like to.

Evidently if we are not burning all the fuel in our charge which is desirable, not only to burn, but also to get rid of, it is useless to add more, and that is precisely what is done if we add either coke or heated blast, as in the one case we are adding carbon which requires oxygen for its combustion—oxygen which we need for another purpose, and in the other we are adding heat, which we should get from the combustion of fuel already in our charge.

The heated blast should, theoretically, only have the effect of increasing the temperature in the focus, and of maintaining the focus near the tuyere level: but owing to the fusible nature of the fuel, this effect is only temporary. The ultimate effect seems to be in the fusing out of the sulphides from a thicker body of silicious matter: the consequent formation of a more silicious slag of a lower formation temperature, resulting in faster driving, lower grade mattes and lower temperature.

There is also more or less loss of air from leakage in the heating apparatus, and the additional pressure necessary to overcome the increased friction due to the greater volume of the heated blast increases both the losses from leakage and from slippage in the blower, resulting in less oxygen going into the focus and less fuel burned."—REDICK R. MOORE.—*London Mining Journal*, Jan. 25, 1908, p. 113. (A. R.)

PRESENT SOURCE AND USE OF VANADIUM.—Vanadium was formerly chiefly derived from certain Spanish lead ores containing 4 or 5 %, but now chiefly from a Peruvian ore containing about 40 % vanadium sulphide, which calcines easily, giving nearly 60 % vanadic oxide (V_2O_5) in the roasted product.

Vanadium is a silvery-white metal of melting point about 2,000° C., and is used to impart certain properties to steels; alloyed with two parts of iron, its melting point is 1,375° C. only, and in this form it is prepared and sold to steel-makers.

A small quantity (0·2 % in engineering steels) greatly increases the strengthening effect of the elements, such as chromium, and also very markedly increases the resistance to fracture by alternating stresses—most important where the metal is subject to vibration.—J. KENT SMITH.—*Bi-Monthly Bulletin of the American Institute of Mining Engineers*, September, 1907, p. 727. (G. H. S.)

PURE COALS AS A BASIS FOR THE COMPARISON OF BITUMINOUS COALS.—In comparing calorific powers of coals, results obtained are calculated on "pure coal," *i.e.*, ash and moisture free coal. The author shows that two other variables, sulphur and water of composition of the ash, should also be taken into account and allowed for.

It is sufficiently close to consider all the sulphur as combined with iron in pyrite and discard it completely, deducting five-eighths of its weight together with the ash and moisture from 100 % to obtain "sulphur-free pure coal": deducting also from the result the heat developed due to the sulphur.

With regard to the second variable, part of the ash at least is fireclay, the chemically combined water of which is lost when the ash is ignited, but not when the coal is dried. This is much more difficult to allow for, as different clays contained combined water from 5 to 12 %, the average probably being about 8 %. But it is probably sufficient to assume that the ash of the coal is similar to the beds overlying and underlying it.

Further, the composition of the pure coal itself is not constant, and coal samples deteriorate on keeping, both of which points must be attended to in making comparisons.—W. F. WHEELER.—*Bi-Monthly Bulletin of the American Institute of Mining Engineers*, Jan. 1908, p. 49. (G. H. S.)

ORE CONCENTRATION.—"The two best friends of the ore producer are fire and water. Water concentration—that is the mechanical washing off of valueless material and concentration of the metallic contents of an ore into a smaller and much more valuable product—has claimed and is claiming the attention of many of the best men we have, but while it has many advantages, it has also its difficulties and limitations. Many ores are complex, their values lying partly with the silica and partly with the various sulphides carried, so that the problem of saving these values is a matter for serious consideration and careful study, and often, even when the theory works out all right, practice declines to conform—and the values 'go down the creek!' Leaving out of consideration the so-called 'surface ores,' that are susceptible to various well-known methods of treatment, suppose we consider the condition of the mine operator who has patiently undergone and overcome the vicissitudes of the earlier work involved in the making of a mine, conceding him a good, strong, well-mineralised vein to start with, and meeting him at the point where his 'free-milling' ore has changed to material that will neither fish nor cut bait—in other words, his amalgamation, water concentration mill (conceding him that also) will not save the values. There are plenty of sulphides—too many to suit him—but all the gold there is in the ore lies with them, what little silica there is being practically or entirely barren. The high percentage of pyrites precludes the possibility of making a marketable product, two or three into one being the very best the tables will do, achieving only about a 60% saving at that—and the ore runs perhaps $5 per ton. Given a prospect plus years of the hardest kind of work, not alone development work, but plenty of blood sweating exertion at the financing end, plus enough free milling material to partly pay expenses during the development period, and then—an almost solid body of low grade sulphides. What does "X" equal? The 'concentrator' men have achieved wonderful results, where the ores are at all suited to water concentration; but every engineer who has any considerable amount of practice knows that in practically all of the developed camps in this country, British Columbia, Mexico and elsewhere, there are thousands of tons of low grade sulphides left underground, because it will not pay to mine them, and the reason it will not is because, in everyday miner's parlance, 'they will not concentrate.' The fact is, that Nature has already concentrated them, in so far as sulphur, iron, more or less copper, perhaps some arsenic, a little antimony, and a sprinkling of lead and zinc is concerned, but the gold is spread out so thin that the mineral, as Nature left it, equals "X" from the miner's standpoint.

Naturally when water concentration is not feasible the mind of the metallurgist turns to fire, and this opens up a field that is certainly full of most alluring possibilities. Matting, the fire concentration of low grade copper ores, has long been successfully practised, but its application to the concentration of iron sulphides, carrying small quantities of gold with perhaps some silver, furnishes a wide opportunity for careful investigation. While it would be an error to say that there are no technical difficulties to interfere with the successful consummation of this process, they are slight when compared with the mechanical and commercial obstacles met with. After the ore supply is assured, the first question that arises is that of available fuel; for while theoretically a straight iron sulphide contains sufficient sulphur to smelt itself, this is not true in practice, although it does admit of a minimum carbon burden and pyritic smelting is to-day being successfully carried on with from 3% to 5% of coke, with ocasionally more or less protracted runs with an even smaller quantity. One of the great advantages of pyritic smelting is the diversity of carbonaceous fuel that may be employed in carrying on the operation. Coke is, of course, preferable, but in the absence of this material, wood, coal or charcoal can be used with excellent results. The next thing to be considered is a sufficient supply of ores carrying at least a small percentage of copper, this being required as a collector of the precious metal contents of the ore to be handled. It is surprising how small a quantity of copper will do this work effectively, practice having demonstrated that 3% is ample, while eminently successful work can be done with as low as half this quantity. There is some diversity of opinion among metallurgists as to whether a hot or a cold blast is preferable in pyritic smelting, but the tendency in practice is in favour of the former, and it is a reasonable hypothesis that oxidation will proceed more rapidly where it is not necessary to utilise an appreciable portion of the heat units at the fusion zone of the furnace in raising the temperature of the air blown in through the tuyeres to the point of effectiveness. As is the case with every business enterprise, the prime requisite for successfully conducting a pyritic smelting plant is good judgment and plain 'horse sense.' The advocates of this process do not claim it to be a panacea, but it certainly does not occupy a position in the world of metallurgy that entitles it to recognition, because it renders available the immense quantities of lean sulphide ores that are susceptible of no other known method of treatment. The mechanical difficulties have been, in a great measure, overcome as the possibilities of the process become more fully appreciated. There is no question but what many plants will be erected in various parts of the country where suitable material is obtainable. Transportation facilities must not be overlooked, because it is outside the province of a pyritic furnace to produce a finished product, transportable in small bulk. A plant of this kind, in a general way, occupies a position corresponding to that of one equipped with concentrating tables where the operation is carried on by water, and its output must be subjected to the further process of refining before its metallic contents are put into shape for the consumer. It has well defined limitations and ores carrying appreciable quantities of lead or zinc are not adapted to this method of treatment; neither is it the object of a strictly pyritic furnace to produce a high grade copper matte in large quantities, but rather to win from the vast bodies of iron sulphides now lying in the stopes of the mines throughout the country their gold and silver values, and thereby materially increase the profit possibilities of many mines that are now partly or altogether idle. To the younger members of the profession, there is no more attractive field for study and research than that of pyritic smelting, because the opportunities therein are practically limitless, and the demands for the services of those skilled in this branch of metallurgy will, within the next few years, be even greater than they are now."—H. P. DICKINSON.—*Mining Science*, Jan. 16, 1908, p. 75. (J. Y.)

EFFECT OF CHILLING ALUMINIUM. — Investigations made by F. G. A. Wilm, of Drewitz, Germany, resulted in the discovery that the strength and ductility of aluminium may be increased if it is chilled in a certain manner. Like steel, aluminium has a critical temperature, which, during the cooling process, is made manifest by the emanation of heat. This is called the point of recalescence. Alloys of aluminium and copper were experimented on. The method of treatment used, however, can be applied to pure aluminium or any other of its alloys. The critical temperature at which the aluminium and copper alloy must be chilled in order to effect this increase in strength and ductility was found to lie between the temperature at which the alloy becomes solid and a temperature not more than 30° C. below the recalescence point. When this temperature is attained the alloy is immediately chilled in cold water. In illustration of this process an alloy of 93% aluminium and 4% copper will be considered. This alloy solidifies at 640° C., and its point of recalescence is 525° C. Taking 30° C. off, we obtain a temperature of 495° C. The alloy has, therefore, to be heated to a temeprature between 495° and 640° before it is chilled. Actual application of this process on an alloy of 96% aluminium and a 4% copper cast in a chill mould resulted in an increase in tensile strength of 48% and an increase in elongation of 2%. Nearly the same increase was found with sand castings, whilst wire, sheet and rod, all produced similar results. Temperatures of recalescence were determined during the investigations for varying percentages of copper in the aluminium. As the copper content is increased from 1 to 33% the recalescence temperature rises from 485° C. to 540° C., the rate of increase of recalescence temperature being less with the high percentages of copper. A patent has been granted in connection with the process, which will doubtless be valued in increasing the tensile strength of aluminium castings."—*Indian Engineering*, Jan. 25, 1908, p. 59. (A. R.)

TWO DETERRENTS TO THE DISSOLUTION OF FREE GOLD IN THE CYANIDE PROCESS.—"I do not know if it has been previously noticed that oil and lime can act as deterrents to the dissolution of free gold by cyanide solution; but owing to the interference with the uniformity of the results of some work in which I was engaged on the Witwatersrand, I was impressed by their possible importance as such, in cases of low extraction that were otherwise inexplicable.

In one pronounced case, when the sand contained 29 gr. of gold per ton no longer amenable to cyanide, it was found possible by laboratory treatment to reduce this residual gold by further cyaniding to 13 gr. per ton; both deterrents being present. The sample containing 29 gr. to the ton was washed with water several times to ensure the absence of dissolved gold, and one portion (*a*) was repeatedly washed with ether, and another portion (*b*) with diluted hydrochloric acid. By evaporation of the washing from (*a*) a substance was recovered resembling vaseline in appearance, and 'touch-lime' (*sic*) was found in the washing from (*b*).

Further treatment with cyanide solution now reduced (*a*) to 15—17 gr. per ton, but (*b*) only to 27 gr. per ton. On treating, after thorough water washing, (*a*) with dilute hydrochloric acid and (*b*) with ether, and again dosing them with cyanide solution (0·25% KCy), both were reduced to 12—13 gr. per ton. The oil film had resisted removal by the working cyanide solution (0·15% to 0·05% KCy) during the treatment period of about six days.

Prolonged aeration by circulation in the sumps was effective in preventing the precipitation of carbonate of lime to any excessive amount in the leaching tanks, and a threat of dismissal for the mill hands on its repetition prevented the further leakage of bearing-oil into the cyanide works."—DUNCAN SIMPSON.—*Institute of Mining and Metallurgy, Bulletin 41*, Feb. 13, 1908. (A. R.)

PLATINISING IRON.—"To platinise iron, says the *Scientific American*, first heat and give it a coat of 22 parts of borate of lead and 4 to 5 parts of blue vitriol mixed with turpentine to paste, washing it off after a while. Platinising formula:—10 parts of platinum, changed into platinic chloride, mixed with 5 parts of ether and exposed to the air until the ether is evaporated. The residue is mixed with 20 parts of borate lead, 11 parts of red lead and some oil of lavender to a paste, which is mixed with 50 parts of amylalcohol. Dip the object in the mixture, dry and heat it."—*Indian Engineering*, Jan. 18, 1908, p. 41. (A. R.)

ESTIMATION OF PHOSPHORUS IN STEEL. — "The results of an investigation by Drs. Frank and Willy Hinrichsen, on the methods of determining the amount of phosphorus present in steel are set forth in *Stahl und Eisen* for February 26. It is shown that the presence of arsenic is liable to influence the estimation of the amount of phosphorus contained in steel by means of ammonium molybdate. The results come out too high. Pure arsenic acid in solution is not precipitated under similar conditions, the arsenic is merely carried down with the phosphorus. The errors caused by the simultaneous precipitation of the arsenic do not exceed under ordinary circumstances (when the contents of the steel in arsenic reach about 0·05%) a total of 0·015%. The simultaneous precipitation of the arsenic is facilitated by the presence of ammonium chloride. It is restricted by the use of free hydrochloric acid, and to some extent also by the employment of an excess of the precipitant, but it cannot be wholly avoided unless the arsenic is first eliminated."—*Times Engineering Supplement*, March 11, 1908. (J. A. W.)

MINING.

APPLIANCES FOR RESCUE WORK.—"The author delivered a lecture on the latest appliances for rescue work in mines. In the first place, he explained that in dealing with any form of breathing apparatus for rescue work, certain requirements must be satisfied. The apparatus must give one a supply of oxygen such as would last at least two hours—a time which would allow of efficient work being carried out—and that supply must be sufficient, not for a man in a resting state but doing the hardest work. In rest a man required something like 0·3 litre of oxygen per minute, but in the hardest work, such as hill climbing, he might use almost 2 litres per minute. The apparatus must therefore allow of a supply of 2 litres of oxygen per minute for two hours. Then the apparatus must have an efficient absorber for the carbonic acid gas given off, and the absorber must keep the percentage of that gas below the amount which was injurious. It was exceedingly dangerous for the percentage of oxygen to fall off much. Normally, it was 21% in the atmosphere, and in the apparatus it ought never to fall below 12%. At 10% there was risk, and at 7% there was very great risk. The dangers of want of oxygen were very insidious, and lack of it might cause loss of consciousness without any warning at all. It was therefore of the greatest danger to have an apparatus so contrived that the oxygen might fall below 12%, and it was of the greatest importance that the man should have warning when the supply of gas was falling off. Then as to the amount of carbonic acid gas which constituted a danger, he remarked that in a badly ventilated room the percentage did not rise above 0·5%, which did not have any injurious effect at all. If, however, carbon dioxide was present to the extent of 3%, it produced some panting and increased depth of respiration. If it rose to 7% it produced unpleasant increased depth of respiration. When it was present to the extent of 10%, there was extraordinary panting, and the whole effort was given up to breathing. With 15% of carbonic acid one might become stupefied and rendered unconscious. Nevertheless, one might live many hours in 15 or 20% without being poisoned. So far as the apparatus was concerned, they should never allow the atmosphere of the breathing bag to rise above 3%, because that amount began to produce troublesome excessive breathing, though it would not do a man any harm. As a matter of fact, it was better to keep it down to about 1%, and then there would be no increased breathing worth talking about. The amount of oxygen, however, was a more important thing than excess of carbonic acid, so long as it did not exceed 3%. Then the breathing dress must be provided with a breathing bag which would allow sufficient air for each breath. It must be ample to allow one to fill one's lungs at each breath. Then there must not be a large dead space, such as a considerable space in a mask would provide, because that would tend to increase the volume of carbonic acid inhaled. Further, the apparatus ought to be as light as possible, and it ought to be so arranged as to incommode a man as little as possible. It should be well disposed about the body. It was most important to well hang the weight, and that was a matter that had been worked out by the German physiologists, in connection with the army, in finding out how to distribute the weight a soldier had to carry about his body. The breathing dress, unfortunately, could not be a light thing, and it was naturally not easy to arrange this apparatus in a light form. The lightest dress on the market was that known as the 'Pneumatogen.' Dr. Hill described the apparatus, and remarked that the supply was quite good under certain conditions. If a man was resting or walking in a quiet way, it would last him one hour, and he could walk a mile and a quarter with it. As long as he was quietly walking it was all right, but supposing he tried to walk quickly, the apparatus could not keep time with it. It was only meant for emergency work, and for use for a short space of time. Prof. Hill then described the 'Dräger' and the 'Shamrock' apparatus, afterwards proceeding to explain the principle of the 'Aerolith' liquid-air apparatus, and also that of the 'Weg.' Finally, he explained the construction and working of the 'Fleuss,' upon which the lecturer had himself effected improvements. The equipment was made, he said, of canvas and leather in one piece, so that the wearer could pick it up and drop it over his head. All he then had to do was to put his mouthpiece in, apply the nose-piece and turn on the oxygen. There were two oxygen cylinders, carried at the back, and a reducing valve, giving a steady supply of 2 litres per minute for two hours. There was an emergency valve and a pressure-gauge carried in front. The caustic potash, used for absorbing the carbonic acid gas, was carried loose at the bottom of the breathing bag, instead of in cartridges. As a consequence, every movement shook the bag and shook the sticks of potash together, with the result that the carbonated surface was rubbed off and the carbon dioxide was the more readily absorbed. It weighed just over 30 lb., the oxygen supply was sufficient, and the carbonic acid was well absorbed even with the severest work."—Prof. LEONARD HILL. —*Colliery Guardian*, Jan. 17, 1908, p. 133. (A. R.)

NEW FUSE FOR INCREASING THE SAFETY OF SHOT-FIRING IN FIERY MINES.—"The fundamental idea embodied in the safety fuse devised by the author is to prevent miss-fires by igniting the blasting charge along its whole length, and not merely at one point, by means of a leaden detonating tube charged with trinitrotoluene. This substance is not very sensitive to external conditions, and is very stable, so that the fuses are safe to handle. The fuse is ignited by a detonator composed of 1 gm. of fulminate, and then detonates at a velocity of about 19,500 ft. per second for the 6 mm. (¼ in.) fuse. The products of detonation include fragments of the leaden tube, and 3·7% of carbon dioxide, 70·5% of carbon monoxide, 1·7% of hydrogen, 19·9% of nitrogen, and 4·2% of carbon. The detonation temperature is 2,428° Cent.

In order to ascertain the risk attending the detonation of the fuse in an atmosphere of firedamp and air, in view of the fact that the detonation temperature is higher than the limit (1,500° Cent.) permitted by the French mining regulations, a number of tests were performed by Messrs. Watteyne and Stassart at Frameries, with a 6 mm. fuse and with one measuring 4 mm., the explosive core in the latter being only 2 mm. in diameter. In the former case it was found that 28 in. (70 cm.) of fuse can be ignited with safety in an explosive atmosphere, ignition of the gas occurring only with a length of 6½ ft.; and as the shorter length is more than sufficient to fire 1 lb. of explosive, it is claimed that the fuse can be used in fiery mines without risk. With the narrow fuse as much as 10 yd. could be ignited without setting fire to the gas, the degree of safety being therefore still greater.

As regards the practical considerations relating to the absence of secondary reactions between the products of an explosion of the fuse and blasting charge, and the complete combustion of the former in the surplus oxygen disengaged by the charge, it results from the fact that the amount of trinitrotoluene is only 2% of the weight of the charge, that in order to compensate the increase in the amount of the hot gases the weight of the limit charge need only be reduced by 2%—a negligible quantity. On the other hand, the combustion of this 2% of trinitrotoluene will not raise the temperature of detonation by more than 150° Cent.; and a similar increase by the oxidation of the weight (10%) of lead present in the tube: a total of 300° Cent. Experiments conducted in shot-holes bored in clay rock showed that when fired along with a charge of ammonium nitrate explosive, the carbon disengaged by the trinitrotoluene is not consumed by the surplus oxygen from the nitrate, but remains as a sooty deposit on the sides of the cavity produced by the explosion through such combustion does occur with certain explosives containing potassium chlorate, the conclusion being that the fuse acts solely as a detonator of ammonium nitrate explosives (grisounite) and that the fuse does not introduce any new element of danger in fiery mines.

Theoretically, the detonating tube presents the following points of superiority over the ordinary method. It enables certain high explosives (such as mixtures of ammonium nitrate and naphthalene or mononitronaphthalene) to be used that have hitherto been unsuitable on account of the difficulty of ensuring satisfactory detonation. It also enables slow powders to act as high explosives, and thereby removes the only obstacle in the way of their use in safety explosives. At the same time there seems to be every probability that the new fuse will enable ammonium nitrate to be used alone as a blasting charge, practical trials having shown that effects equal to those of grisounite can be obtained by this means, especially with a fuse 7½ mm. thick, seeing that this fuse does not burn in the charge of nitrate.

The author also proposes to apply the new fuse to nitroglycerine explosives. In this case the fuse will be laid on one side of the shot hole so as to project a short distance from the mouth, the cartridge being then introduced singly in the ordinary way and stemmed. The detonator will be attached to the end of the fuse outside the hole; and with a narrow fuse (say, ⅕ in. thick) there would be no risk in fiery mines, in view of the results already mentioned as having been obtained in the tests at Frameries, the system of attaching the fuse to the detonator increasing the safety of the method. It is admitted, however, that the arrangement will require to be properly tested before being admitted into mining practice. Another advantage claimed is that, the detonator being outside the shot-hole, it can be replaced, with less risk in the event of a miss-fire."—M. Lheure, *Annales des Mines.—Colliery Guardian*, Jan. 24, 1908, p. 165. (A. R.)

Borehole Temperatures.—"A meeting of the Edinburgh Geological Society was held recently, when Mr. James Grant Wilson, of H.M. Geological Survey, exhibited a diagram prepared from the journal of the Balfour bore in Fifeshire. The bore began near the top of the millstone grit and passed through the upper limestone group, the middle coal-bearing group, and stopped about the horizon of the Hozie limestones in the lower limestone group. The remarkable feature of this bore is the great development of the coal in the carboniferous limestone series. Mr. E. M. Anderson, H.M. Geological Survey, described the temperature observations taken by Mr. John Gemmell, mining engineer, in the Balfour bore. One maximum thermometer by Cassella was used, and four by Hume on the principle of a clinical thermometer. The greatest depth which could be reached was 3,955 ft., and at this depth a temperature of 92·2° F. was recorded. This corresponds to a temperature gradient of 1° F. in 87·5 ft., which is well under the average. The gradient was least in the upper part—a feature which was accounted for by the high conductivity of strata (including millstone grit), passed through in this part of the bore."—*Colliery Guardian*, Jan. 24, 1908, p. 182. (A. R.)

Points in Breathing Apparatus.—"All breathing apparatus for use in mines must be airtight, as far as the passage of air from the outside to the inside is concerned. Such apparatus should be fitted out with emphatic mechanical warning instruments and will warn the wearer of the failure of oxygen before the percentage has fallen to a dangerous limit. The result is best obtained by arranging and working the apparatus so that it becomes filled with a mixture containing 60% of oxygen. Before the oxygen percentage has been reduced to the limit of safety the apparatus will then become so empty and respiration so impeded or impossible that the attention of the wearer cannot fail to be attracted mechanically. Any apparatus which permits the possibility of the user being unknowingly led into breathing a mixture containing less than 12% of oxygen must be looked upon as extremely dangerous. All breathing apparatus should also have attached a moderately distended bag to act as reservoir and as a buffer or cushion for the ebb and flow of inspiration or expiration may be contained in it quite easily. The dead space in a respiration apparatus should not exceed 200 or 250 cubic centimetres." (T. L. C.)

Helmets for Miners.—"Many ways of decreasing the injurious effects of dust on miners have been tried. In these methods either the dust is allayed by spraying or the dust is prevented from entering the lungs of the miners. To the water spray miners often object because of the difficulty at many dry mines of getting water for the supply tank. To the respirators the miners object mainly because it hampers conversation with a nearby partner. At the St. John del Rey Mine, Brazil, helmets supplied with air from the compressed air pipe are worn by the machine men. The small rubber pipe, which brings the air to the helmet, can be detached instantly by means of a connection, the valve in which closes automatically when detached. Expansion cools the air, and renders work in a hot stope much pleasanter." (T. L. C.)

MISCELLANEOUS.

Earthquakes and Petroleum Fields. "In a paper recently presented to the French Academy of Sciences Mr. Tassart points out an interesting connection between petroleum layers and seismical phenomena.

A thorough examination of the location of petroleum fields has brought out the following facts:

1. All petroleum fields situated in recent strata are confined within regions of maximum seismical activity or in their immediate vicinity.
2. Petroleum fields are found only exceptionally in ancient strata, with seismical zones.
3. Those petroleum fields which are situated outside of such zones belong to ancient strata and were formerly the seat of a lively seismical activity."—*Indian and Eastern Engineer*, Jan., 1908. (A. R.)

Zinc Dust in Cement.—"It has been known for some time that a cement made out of certain oils and zinc dust possesses the useful property of becoming exceedingly firm and adhering closely to iron, steel and other metals when heated to a temperature of 150° C., or even less, if treated for a sufficient length of time. The theory of the nature of zinc dust which Mr. Alfred Sang has given in his article on sherardising in our last issue, explains this hardening by the release of the zinc from its peculiar condition, to form a solid without the formality of passing through the liquid stage. In the body of the cement it is protected from oxidation and the zinc becomes mechanically continuous. It is an excellent composition for caulking cracks and crevices in metallic objects, for packing joints and for smoothing off the surface of castings."—*Electro-Chemical and Metallurgical Industry*, June, 1907, p. 248. (K. L. G.)

Synthesis of Gems.—"In a recent German patent (D.R.-P., 182,200) a method is described for producing artificially certain mineral species, such as olivine, zircon, beryll, and spinel, in a definitely crystalline form. This consists in dissolving their constituent oxides in appropriate proportions in molten sodium or potassium metaborate. The temperature of the fused mixture is then raised to 1,300° C., when the alkali metaborate volatilises, leaving the artificial mineral in the form of crystals. In addition to obtaining products corresponding with the naturally occurring minerals, other compounds having no mineralogical counterparts may be produced by this method. A nickel chrome spinelle, NiO, Cr_2O_3, prepared from its constituent oxides, was obtained in the form of small green crystals. It is well known that the mineral species corundum occurs in the form of very variously coloured stones ranging from colourless sapphire to brown and opaque corundum. Between these extremes one meets with sapphires tinted in yellow, green, blue, red, and intermediate hues, and it is generally supposed that these various colours are due to the presence of compounds of iron, manganese, chromium titanium, or other foreign elements. In the *Comptes rendus de l'Académie des Sciences* F. Bordas describes experiments in which the colour of these gems is caused to change by exposing the stones to the action of a very radioactive specimen of radium bromide. In these circumstances a blue sapphire assumes successively green, light yellow, and dark yellow tints, whilst a red sapphire develops in succession shades of violet, blue, green, and yellow. These experiments justify the belief that the distinctive colours of these precious stones are not due to the presence of any particular oxides. The variation always occurs in the above sense from red to yellow, and it seems likely that the topaz represents the last term in this transformation. Moreover it seems probable that these gems are found in regions where the surrounding soil has a certain degree of radioactivity. This idea receives support from the fact that the yellow sapphires are the commonest, although yellow and blue sapphires frequently occur together. The gems which have been thus artificially coloured are not radioactive; they do not become luminescent in the dark under the influence of radium bromide, but they retain their colour on heating."—*Times Engineering Supplement*, Jan. 1, 1908. (J. A. W.)

A MODERN METHOD OF LEAD BURNING.—"The use of pure oxygen in place of air not only enables much more perfect combustion and higher temperatures to be obtained than are possible with the hydrogen and air system, but it possesses a still further economic advantage in that either coal or natural gas from the town supply may be employed instead of hydrogen.

The oxy-coal gas blowpipe is constructed on the injector principle, and is now extensively used by lead burners. By its use oxygen delivered under slight pressure from a trade cylinder is caused to draw the gas direct from the ordinary town supply and then eject the mixed gases in the right proportion through the nozzle of the blowpipe. The system is being introduced by the Linde Air Products Co., of Buffalo, and its superiority over the old hydrogen and air system, as regards quality of work, economy and convenience in use is admitted by all who have adopted it. It is suitable for ordinary flat work, horizontal and upright joints, overhead patching and the jointing of ordinary lead piping. Any workman conversant with the hydrogen and air system of lead burning will experience no difficulty in operating the oxy-coal gas blowpipe.

It is estimated that in regard to the question of economy there is a saving of approximately 50% by using the oxy-coal gas system in preference to the old hydrogen-air method of lead burning. There are besides many other advantages, as enumerated here below:

1. The hydrogen generator is dispensed with.
2. The air bellows or pump is dispensed with, and consequently the services of a boy are not required.
3. Instead of having to move a heavy hydrogen generator and bellows from one job to another, it is generally only necessary to move a light cylinder containing the oxygen required.
4. No apparatus to get out of order, involving expenditure, delays and repairs.
5. As neither zinc nor sulphuric acid is used, there is no deleterious matter to be carried through the blowpipe to act injuriously on the lead seam.
6. No preheater or 'Fou-Fou' is required on heavy work. The oxy-coal gas flame is so hot that even heavy lead in wet and cold positions can be burned *in situ* without preheating.
7. No gas is generated when the blowpipe is not in use, and consequently there is no waste of gas and no charge to withdraw overnight.

It may be stated that for cases where a supply of town's gas is not available coal gas can also be obtained in cylinders, or the oxygen may be used advantageously with the hydrogen 'machine.'"—*Electro-Chemical and Metallurgical Industry*, Feb., 1908, p. 82. (K. L. G.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

LEAD REFINING BY ELECTROLYSIS. By ANSON GARDNER BETTS. 394 pp., $4. (New York: J. Wiley & Sons.)

"This book contains information resulting from years of study, experiment, and practical work of the electrolytic refining of lead bullion, which is now an established metallurgical process and one from which further developments may be expected in the future. The methods of slime treatment are also discussed in considerable detail."—*Mining and Scientific Press*, April 18, 1908, p. 538. (W. A. C.)

MINE GASES AND EXPLOSIONS. By J. T. BEARD, C.E., E.M. 402 pp. price $3. (New York City: John Wiley & Sons.)

"The scope of the book is shown by the following chapter headings: The Chemistry and Physics of Gases; Heat and its Effects; The Atmosphere; The Common Mine Gases; Mine Explosions; Safety Lamps; Testing for Gas. The subjects treated are particularly pertinent at the present time.

The book is interestingly written and contains much valuable information. The subject of dust explosions is clearly treated, a good résumé of much of the literature upon this subject to the present time being given. The treatment of safety lamps is probably the most complete treatment of the subject thus far published. There are some points in the arrangement of the book in which many will differ with the author—for instance, since the title of the book is Mine Gases, under Specific Gravity, one naturally expects to find something about the specific gravity of gases, but no mention whatever is made of the subject at this place, while the subject of the specific gravity of gases is treated under the head of the Behaviour of Mine Gases. The tables are frequent and interesting, but their value and also many of the incidental references to other works would be increased if the original source of the information were stated, so that those desiring to do so could consult the original articles. Under the heading Mixture of Gases and Air, the author gives the name Flashdamp to what he considers a more or less definite combination of marsh gas and carbon dioxide, and holds that this mixture is more dangerous than firedamp. This seems rather unreasonable and is opposed to the author's own statement on p. 104, where under the heading of Effects of Other Gases on Firedamp, he says: 'Carbon dioxide reduces the explosiveness of firedamp in proportion to the amount present in the mixture. When the firedamp is at its most explosive point, one-seventh of its volume of carbon dioxide will render the mixture non-explosive.' The percentage composition of flashdamp is given as 38% carbon dioxide and 62% methane. If the statement quoted is true, which it undoubtedly is, it is difficult to see how a mixture of practically one part CO_2 and two parts methane can be more dangerous than pure methane. Inquiries made of a large number of mining men do not substantiate the occurrence of any such definite mixture.

The author also gives a number of tables to show that there is a direct connection between volcanic and other earth movements which will account for periods of an unusual number of explosions. We doubt very much if this contention can hold, and certainly all of the other writings upon the subject oppose it, and it seems contrary to geological acts."—*Mines and Minerals*, March, 1908, p. 373. (A. McA. J.)

SIMPLE MINE ACCOUNTING. By DAVID WALLACE. 64 pp., $1. (New York: Published by the Hill Publishing Co.)

"This book is intended for those who have no chance to learn book-keeping. It is well adapted for the use of foremen and superintendents, being sensible and simple. It covers the ground satisfactorily, from vouchers, cash-books, and ledgers to stock and smelter records." — *Mining and Scientific Press*, April 18, 1908, p. 538. (W. A. C.)

METHODS AND DEVICES FOR BACTERIAL TREATMENT OF SEWAGE. By W. M. VENABLES. 236 pp, $3. (New York: J. Wiley & Sons.)

"This subject is fully treated from the viewpoint of the civil engineer, and contains a full summary of the known engineering principles involved in the subject under treatment and a statement of the problems unsolved." — *Mining and Scientific Press*, April 18, 1908, p. 538. (W. A. C.)

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Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) means provisional specification, and (C) complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)

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